Metallurgical and Chemical Engineering

A Consolidation of ELECTROCHEMICAL AND METALLURGICAL INDUSTRY and IRON AND STEEL MAGAZINE

NEW YORK, JULY, 1910.

CONTENTS.

Dictionary of Metallurgical and Chemical Material.

At about the same time that this issue reaches our readers, they will receive the 1910 edition of our Dictionary of Metallurgical and Chemical Material. The first edition of this dictionary, issued last year, has found such a friendly welcome as to convince us of its usefulness for the many users and buyers of chemical machinery, appliances and material. The second edition has been completely revised and considerably enlarged. Our thanks are due to many friends for corrections and suggestions, all of which have found due consideration. For the future we ask again for the co-operation, by criticism or suggestion, of all who will use the little book in the task, often so troublesome, of looking for information "where to get things."

Bureau of Mines.

The final establishment of a Bureau of Mines in the Department of the Interior has occasioned some inquiry as to the relation of the Bureau to the metallurgical industry. Briefly, there is no provision for any work or investigation that would be of interest to metallurgists, and any such provision must be a development of the future. As finally approved on May 16, 1910, the bill provides for the establishment of the Bureau and for the appointment of a director "who shall be thoroughly equipped for the duties of said office by technical education and experience." The original appropriation suggested to carry out the work of the Bureau was $546,050, and this included an appropriation of $150,000 for the testing of structural materials, clays, cements, etc. The latter was, however, taken out of the bill in the House, and that phase of the work transferred to the Bureau of Standards. Unless it is restored in the Senate and finally agreed upon, the testing of materials will not be a function of the Bureau of Mines.

* * *

Practically speaking, the work of the Bureau of Mines will, for the present, be a continuation of the work of the technologic branch of the Geological Survey. An appropriation of $100,000 has been made for "the analyzing, testing and treatment of coals, lignites, ores, and other mineral fuel substances," which is one phase of the work that is of interest to engineers, and which was discontinued under the Geological Survey on account of lack of funds. Further appropriation of $310,000 is asked for the investigation of mine explosions, the safety of miners, the improvement of the conditions under which mining is carried on, the use of explosives and the prevention of accidents. This will also provide for the establishment of rescue stations where apparatus useful in entering mines in which disasters have occurred will be kept for use, and where experts in the use of these appliances will have headquarters for instructional purposes, and to render aid when called upon. Thus it will be seen that the coal-mining industry is the first and greatest beneficiary of the Bureau, and that the interests of metallurgy in the Bureau will depend on future developments.

Carbon Cells.

As the idea to devise a "carbon cell" continues to occupy the minds of inventors and investigators, in spite of the many failures of former attempts, a brief review of what is being attempted and what has actually been accomplished should not be out of place. What is the object of a carbon cell? We can make this clear by the perhaps unfamiliar yet striking analogy with Mr. Edison's iron-nickel cell. In the Edison accumulator the reaction during discharge is the reduction of a higher oxide of nickel to a lower oxide on one side and the oxidation of iron on the other side; the electrolyte, a solution of potassium hydroxide, simply serves as the carrier of the ions, so that the electrolytic reaction may go on, but the electrolyte itself is not changed at all, in the whole, during discharge. This particular nickel-iron accumulator is a special and the most successful example of a general type of cell to which the name "oxygen-lift cell" has been given. If M and R are the materials of the two electrodes, the general equation of the discharge of an oxygen-lift cell is $MO + R = M + RO$; during charge the reaction is reversed. Now, the ideal carbon cell, as it should be, is a special type of oxygen-lift cell; namely, a cell in which the material R is carbon. Its equation is $2MO + C = 2M + CO_2$. The problem is to find a suitable electrolyte and a suitable material M.

* * *

While the Edison cell is an accumulator to be discharged and recharged periodically, the carbon cell is essentially intended to be a primary cell for continuous discharge. That is, when the material MO is reduced to M during the discharge of the cell, the intention is to reoxidize the M continuously at once by means of new oxygen from the air. Such a cell will give out continually electrical energy, according to the free energy of the reaction $2MO + C = 2M + CO_2$, at the expense of carbon and atmospheric oxygen which combine to carbon dioxide. But in contradistinction to the combination of carbon and oxygen by direct combustion, the scheme of the carbon cell requires that the oxygen is not brought into contact with the carbon, but that the fresh supply of oxygen is made to the other electrode, the M electrode. The energy of combination of carbon and oxygen by direct combustion appears in form of heat; in electrochemical combination it appears directly as electrical energy. Hence the necessity of producing electrical energy by the roundabout way using steam engines and electric generators is avoided, with its necessarily very low efficiency, limited by the second principle of thermodynamics. We are not limited in this way in the carbon cell; in principle we could transform the whole free energy of the reaction into electrical energy. We thus would get several times the amount of electrical energy from a ton of carbon which we get now by roundabout methods. That this would mean an industrial revolution there can be no doubt.

* * *

The attractiveness of the proposition is evident. The scheme is undoubtedly sound in principle and theory. But to devise a theoretically sound scheme of an electrolytic reaction and to build in practice a cell which operates according to such a scheme are two very different things. Cells have been built which have been believed to realize more or less the scheme outlined above, and they have been called carbon cells and have been heralded as the dawn of a new civilization. But detailed investigation has always shown later on that they were not true carbon cells and nothing came of them. In two of the most notable examples fused sodium hydroxide and concentrated sulphuric acid have been used as electrolyte respectively, the former in the widely discussed Jacques cell, consisting of carbon and iron covered with oxide, in fused sodium hydroxide. Haber has shown in a paper before the International Electrical Congress in St. Louis, and his results have later been confirmed, that what primarily happens is a direct chemical reaction between carbon and sodium hydroxide, giving sodium carbonate and hydrogen; the cell then acts simply as a hydrogen-oxygen gas cell; the carbon acting as a "hydrogen electrode" and the iron as an "oxygen electrode." The reaction is, therefore, entirely different from the desired reaction of the imaginary carbon cell. The cell is of no practical value, because the expensive carbon electrode and the expensive sodium hydrate are changed into cheap soda, only to yield a little hydrogen which acts electromotively with the oxygen at the other end.

* * *

Junger has later built a "carbon cell" with concentrated sulphuric acid as electrolyte. Recently Taitelbaum has made quite extensive and interesting experiments along similar lines, in which he found vanadium salts to be particularly suitable in connection with the material M. Without entering here into a discussion of these researches, we will simply emphasize that here again the carbon electrode has been found not to act in the way desired. Hot concentrated sulphuric acid dissolves carbon, oxidizing it with formation of sulphur dioxide. The carbon electrode then acts as a sulphur dioxide gas electrode; and the desired carbon cell is in reality a sulphur dioxide-oxygen gas cell. Thus the result is again negative, as before. The difficulty would seem to be to find a suitable electrolyte which, just as in the Edison cell, acts simply as a carrier of the ions. So far it has not been found for the carbon cell. But this fact does not prove the impossibility of the scheme.

A Disadvantage of Reinforced Concrete Buildings.

The undeniable and important advantages of reinforced concrete for industrial buildings are now so widely recognized that we may call attention to the fact that under the peculiar conditions of industrial development in our times one of the strongest points of reinforced concrete may become a disadvantage—that is, its practical indestructibility. We no longer pretend to build "for eternity." We know we live in a transition era and the idea of scrapping good things for the sake of better ones has lost all terror for us and has even become familiar. In view of the constantly changing character of our industries, the lack of flexibility of a reinforced concrete structure, the difficulty of tearing it down may occasionally prove embarrassing; and we have an actual case from chemical engineering practice in mind in making these remarks.

Treatment of Slime.

More and more we tend to believe in fine grinding in the treatment of ores bearing precious metals, and the improved methods of handling slime successfully are enabling us to make better extractions, so that the percentage recovery of

gold and silver from an ore amenable, say, to the cyanide process, should be relatively high. On the other hand, the slime formed in the concentration of copper, lead, zinc and tin ores is the source of a great deal of trouble, and its treatment constitutes a problem that is not near solution in the customary mechanical concentration processes. While some of the devices in use produce high-grade concentrates in the final treatment of slime, they also have the defect of making high tailings. In lead and copper concentrating is this especially true, the vanner being the final concentrating machine, with retreatment of the tailings, in some instances, on buddles. Elsewhere in this issue our readers will find a paper by Mr. Walter McDermott which is a valuable contribution to the literature of slime concentration. It clarifies the atmosphere surrounding this problem and presents certain conclusions that are deduced from a consideration of the progress in concentrating methods during the past 30 years.

* * *

The author intentionally leaves out of consideration those methods which are based on principles other than gravitation. Looking into the future he believes that an advance will be made only by a concentrator which embodies a large and smooth surface, a regular settling motion which shall not jar the finest particles from the surface when settled, and the production of a clean concentrate without a middling. The question may be raised whether it is necessary to look for such a mechanical concentrator, when there are other forces which can be utilized to save fine mineral from slime. The various flotation schemes, which make use of the property of surface tension of the mineral particles, seem to meet with success, and suggest that perhaps the solution of the problem of slime concentration is to be found elsewhere than in mechanical concentration. The attempts to employ centrifugal force have not been very successful, although the idea is attractive; but in view of the fact that an extended trial of a centrifugal slime concentrator is soon to be made on copper ore slime by a large Western company, the last word cannot yet be spoken on this class of mechanism.

Available Iron Ores.

Two publications of widely different character bearing upon the iron ore resources of the world have recently appeared. One is a couple of ponderous volumes containing more than 1000 pages, with an atlas of 43 maps embodying the results of an enquiry edited by Mr. J. G. Anderson, general secretary of the International Geological Congress, which will meet at Stockholm later in the year, and the other is a popular article in the *Atlantic Monthly* by Dr. Henry M. Howe. The report for the Geological Congress aims to cover all that is known of ore deposits the world over, and tabulates the deposits, both as to tonnage of ore, as such, and as to tonnage of iron which may be regained therefrom. There is a great maze of figures, by far the worst we have ever encountered on this subject, and clearing away a mass of almost infinite detail, we reduce the figures to the lowest terms, to represent the tons of iron which may be regained:

Actual ore supplies....................$20 \times 10^9 +$
Potential ore supplies.................$10 \times 10^9 +$

Dr. Howe's article is intended to be a popular one. If Wall Street had not taken a hold upon the iron industry, some years since it would not have been necessary for the Doctor to write the article. He tells the public what it needs to be told, that of iron ore there is no end, and that it is only a question of development of processes whereby apparently negligible amounts of iron may be regained. In Wall Street there was promoted the idea that there was no ore but Lake Superior ore. It was easier to frame up that idea than to explain that there was plenty more iron, but it would be much more expensive to isolate it. The question of how much available ore there is in the world is as indefinite a one as is the question what is the volume of a cone with a very obtuse point, neither the angle at the point nor the distance from the point at which the plane is to be drawn being given. Dr. Howe does not need to bother with the angle, for he puts the plane at what amounts to an infinite distance when he suggests that the earth itself is iron. With that conception we are not sure there is left anything for us mortals to stand on, since it is not certain that in gaining the iron there will be any slag produced!

* * *

These presentations are useful in their way, the one as an aid in the study of geology, the other to correct a popular misapprehension. The metallurgical world is not concerned with either, but with the relatively minute—from a mathematical standpoint—question of drawing the plane, very near the apex of the cone in the present state of the art, between the commercially available ores and those not commercially available. That successive generations will be able to draw the plane farther and farther down in the cone, moving it more rapidly than the wants of commerce expand, is perfectly obvious. The world has made, approximately, one billion tons of iron. All the knowledge we have now has been acquired during the manufacture of that tonnage. The ores at present easily available will give us, with no advance in methods of treatment, many times as much iron, but—how much more shall we know when we have made another billion tons? Along mechanical, rather than metallurgical, lines, a great deal of progress has been necessitated by the needs already encountered. The ores from which, centuries ago, wonderful grades of iron and steel were made by the Catalan forge, are not available to-day in commercial quantities, but there is still an enormous tonnage of ore which can be treated by the processes of a quarter or a half century ago. Along mechanical lines the Mesabi range has been the field of wonderful progress. Two decades ago the presence of ore there was not suspected. Suddenly large deposits were found, almost underneath the sod. Had it been necessary to strip 100 ft., or even 50 ft., of overburden to gain that ore it would have awaited a more convenient time, but to-day there are several stripping operations involving the removal of more than 100 ft. of overburden. Enormous investments are required, for, as a rule, a 40-acre or 80-acre tract will be stripped before any ore is mined. Thus capital is easily found for such advances as may be required. For the much more intricate problems of the future, involving the regaining of iron from very low-grade ores, the capital will no doubt be found, and the engineers and metallurgists will welcome the problems they will be asked to solve.

Niagara Falls Meeting of the American Institute of Chemical Engineers.

The second semi-annual meeting held at the Clifton Hotel, Niagara Falls, Canada, from June 22 to 25, was very successful. The weather was ideal and the attendance satisfactory.

In the opening session on Wednesday, June 22, the Institute was welcomed by Mr. A. Monroe Grier, K. C., of the Canadian Niagara Power Company, in a witty address in which he emphasized the ideal conditions of the locality for both work and play. The membership committee reported an increase of 17 for the past six months. The committee on medal reported specific plans for a permanent foundation for the awarding of a medal annually for the best paper presented during the year to the Institute. This medal will first be awarded at the next semi-annual meeting.

Action toward incorporation of the Institute was taken, to be consummated at the annual meeting. This will be held in New York during the early part of December. The amendment to the constitution to make ex-presidents ex-officio members of the Council for two years was favorably considered.

Mr. Edward R. Taylor, of Penn Yan, in a paper on The Changes in Industrial Chemistry Caused by Electricity called attention to the very extensive use which may be made of electricity in our municipal life, for instance, for water and air purification and sewage disposal, as well as an aid to agriculture, installations even on a small scale being feasible. Many ways were pointed out in which strictly chemical industries could be economically carried out by means of electricity.

A paper on the Corrosion of Iron and Steel and Its Prevention, by Mr. G. W. Thompson, of New York, read by Mr. Andrew Moyer, Jr., of New York, gave a summary of what is known at present on this important subject. As a general rule it was claimed that corrosion is not dependent on the purity of the steel so that the use of an expensive pure steel is not warranted. All the known causes of corrosion were given and methods of prevention in each case were outlined. Specific rules on cleaning of the steel and the paint to be applied were given.

Dr. F. G. Wiechman read a paper on "Protal: a new product for use in the arts." It is a new plastic which has many of the valuable properties of rubber and can be substituted for it in all cases where either soft or hard rubber is required. It can be moulded into any form and has the very decided advantage that protal does not deteriorate on account of oxidation. It is made from vegetable albumen, which for this purpose is best obtained from the seed of a palm which can be produced in any quantity. These seeds are used for the manufacture of buttons in which there is very great waste, which can be used in the manufacture of protal.

Protal can be compounded with rubber and a large variety of other resinous substances. Among these the most important is bakelite making protal-bakelite which is absolutely waterproof and unaffected by both cold and hot water. It resists the strong oxidizing influence of exposure for three months to strongly oxidized air.

Wednesday afternoon an excursion was made to the plant of the International Paper Company in Niagara Falls, N. Y. At this plant the wood mill was visited for both mechanical wood pulp and sulphite pulp, as well as the paper mill in which ordinary newspaper is made. The capacity of the mill is 110 cords of wood per day.

The plants of the Canadian Niagara Power Company and of the Ontario Power Company were then visited.

The address of the president, Dr. Charles F. McKenna, on The Study of Materials as a Subject in a Course of Chemical Engineering was delivered on Wednesday evening. Dr. McKenna emphasized the great importance of a more careful study of the properties of materials. A masterly discussion of the principles of the inspection and testing of materials under practical conditions was given. A more detailed account

of this address is reserved for the July issue of this journal.

The report of the committee on Chemical Engineering Education was then read by Dr. F. W. Frerichs. The committee reported that the ordinary four years' course in chemical engineering as given in American universities and technical schools does not give sufficient training for technical work, but that at least five years' study would be necessary. The committee was also of the opinion that the courses offered in European technical schools were superior to those offered in American schools. The teaching staff of American schools is often too limited in number and deficient in practical experience.

The entire second day, Thursday, was devoted to visits to several important industries of Buffalo. A special car was chartered to take the members of the Institute and their guests from the hotel to the various works and return to the hotel. The attendance on these excursions was about 45.
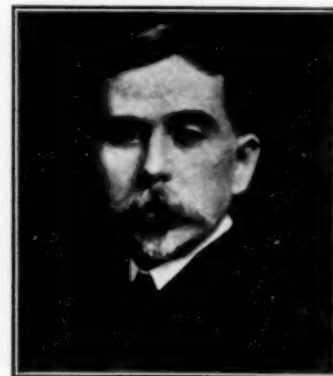
CHARLES F. M'KENNA, PRESIDENT AMERICAN INSTITUTE OF CHEMICAL ENGINEERS.

At the plant of the Lackawanna Steel Company the large ore handling apparatus was first inspected. The blast furnaces and Bessemer converters, and the very economical system of utilization of the blast furnace gases proved very interesting. The hot gases were cooled by passing through the stoves used for heating the air blast, after which the dust was removed. The gases were then burned under boilers or utilized to drive gas engines of 2000 hp. The open-hearth furnaces were also visited.

At the plant of the Larkin Company a complimentary luncheon was served to the visitors, after which a portion of the plant was inspected. The great variety of operations carried out and the excellent organization of the plant was remarked by all.

The last plant visited was that of the Linde Air Products Company. Here the visitors found air liquefied on a large scale in a very well-designed plant. The nitrogen is allowed to evaporate leaving oxygen of 96 to 99 per cent purity. This is charged into cylinders and sold as compressed oxygen. About half of the oxygen of the air liquefied is obtained. The cold gases coming from the liquefied air are used to cool the compressed air before liquefaction.

On Thursday evening a dinner was held at Hotel Clifton. The president, Dr. Chas. F. McKenna, acted as toastmaster in his usual tactful and charming manner. Mr. J. A. DeCew, of Montreal, Canada, welcomed the Institute to Canada.

Dr. Edward G. Acheson said that as he was a resident of Canada and a citizen of the United States he felt that he also desired to welcome the Institute. He expressed the opinion that in the name "Chemical Engineer" the Institute had selected a name implying a profession which studied matter and its transformation both on a small and large scale and so was more fundamental in scope than any other profession.

M. de Chatelain, professor of the Polytechnic Institute of Russia and president of the Electrolytic Section of the Imperial Technical Society of Russia, said that he found America very much in advance of Russia in the application of electricity. He had come to America in order to study our methods of harnessing and utilizing water-powers for the generation of electricity.

Past-President Samuel P. Sadtler gave an outline of the growth of the Institute during the past six months.

Mr. A. Monroe Grier, K.C., of the Ontario Power Company, then made an eloquent address on the future of electricity.

On Friday morning the remarkable growth of the chemical industries of Canada was outlined in a paper by J. A. DeCew, of Montreal, Canada. The greatest growth was shown to have taken place in the sugar industry and the distillation of wood and the utilization of the products.

A very comprehensive paper was read by Dr. Oscar Snider, of the Western Electric Company, Chicago, Ill., on the manufacture and industrial application of Ozone.

The most recent development in this modern industry was given. The use of low voltages was declared to give a low yield as well as a low concentration. At least 10,000 volts must be used, while voltages as high as 40,000 are being used.

The application of ozone to the purification of water and especially of contaminated air was discussed at length. The use of ozone for the sterilization of a great variety of substances, as well as food products and as an oxidizing agent in chemical industries, was explained. Among these are the preservation of eggs and milk, bleaching, the cyanide method of extracting gold and the production of ferrocyanides.

"Some Problems in Chemical Industry," giving some theories of chemical action, was then read by J. T. Baker. Dr. Ralph E. Meyer, of the Westinghouse Lamp Company, then presented a paper on commercial manipulation of refractory elements for incandescent-lamp purposes. The efficiency of the various materials for lamp purposes was discussed. Tungsten has been found to be the most practical material for this purpose. The method of testing tungsten ore for the extraction of the metal was given, as well as the method of forming the filaments for the lamps. In a similar manner the manipulation and properties of metallic molybdenum were described. This metal is well adapted for use as filament supports. The new method of using wire structure for lamps was explained. Lamps of this type having a candle-power varying from 12 cp to 500 cp were shown. Samples of all lamp materials in various stages of manufacture were shown.

On Friday afternoon the manufacture of graphite, aquadag and oildag at the plant of the Acheson Graphite Company was inspected. The remarkable manner in which the element carbon has been modified to suit a great variety of purposes was very suggestive of what can possibly be done with other elements.

At the evening session the papers by A. Bement, of Chicago, on the changes in coal due to storage; by Schuyler Frazier, on nitric and mixed acids, and by Dr. Wm. M. Grosvenor, on plant design, were read. Mr. Bement's paper gave the results of tests on coal stoves for three years.

Mr. Schuyler's and Dr. Grosvenor's papers were very suggestive and led to an interesting discussion. A very comprehensive paper on the subject "Underground waters for manufacturing purposes," by Wm. M. Booth, was read by title.

The total attendance at the meeting was about 75. The attendance at all the sessions and excursions was excellent and a marked increase in interest was shown as the meeting progressed.

San Francisco Meeting of the American Chemical Society.

The complete program, just issued, shows that the arrangements for the San Francisco meeting of the American Chemical Society have been made with utmost care. There can be no doubt that also the trip with the special train, which leaves Chicago on July 4 and arrives in San Francisco on July 12, will be greatly enjoyed by all who will participate. Headquarters in San Francisco will be at the St. Francis Hotel.

At the first session, to be held on July 13, the president, Dr. Wilder D. Bancroft, will present an illustrated lecture on positive photography. Papers will be presented by Dr. E. C. Franklin on liquid ammonia as a solvent and the ammonia system of acids, bases and salts; by Dr. W. F. Hillebrand on chemistry in the Bureau of Standards, and by Dr. H. E. Barnard on the use of sodium benzoate as a preservative of food.

In the afternoon an excursion will be made on the Ocean Shore Railway to Half Moon Bay and Lunitas. In the evening a smoker will be held at the Fairmount Hotel, where there will be a formal dinner for ladies at the St. Francis Hotel.

On Thursday, July 14, in the morning, sessions of the Industrial, Physical and Inorganic, and Organic Divisions will be held at the St. Francis Hotel. At this meeting the Industrial Division will hold its symposium on smelter smoke. This will be opened by the following three papers:

Mr. Charles Baskerville. "The Smoke Problem and the Community."

Mr. W. E. Ebaugh. "The Neutralization and Filtration of Smelter Smoke."

Mr. F. G. Cottrell. "The Electrical Precipitation of Suspended Matter."

Another paper has been promised by Mr. Edmond O'Neill. This will be followed by a general discussion.

Other sessions of the different divisions will be held in the mornings of July 15 and July 16.

Excursions have been arranged for the afternoon of Thursday, July 14, to Mount Tamalpias and the Muir woods; for Friday afternoon to the Selby Smelting & Lead Company, and a steamer trip around the Bay of San Francisco and out through the Golden Gate; for Sunday, July 17, to the Italian-Swiss colonies at Asti, and for Monday, July 18, an excursion on the Sacramento River.

For the entertainment of those who choose to return by way of Seattle, the Puget Sound Section has arranged a very attractive program.

As has already been noticed, the special train which is to take the Eastern members to San Francisco, will run from Chicago (July 4), via Colorado Springs, Grand Canyon of Arizona, Redlands, Riverside, Los Angeles and Santa Barbara; stopping at Adamana for a trip to the petrified forest and at Lanz for a visit to the borax mines. This train will arrive at San Francisco in the morning of July 12.

Dr. Wilder D. Bancroft is the president and Dr. Charles L. Parsons, Durham, N. H., is the secretary of the American Chemical Society. Mr. Ralph A. Gould is the chairman of the executive committee of the local arrangements committees for the San Francisco meeting. He will meet the party on the special train at the Grand Canyon as the representative of the California Section and will accompany them as they enter the State and until they reach San Francisco.

Iron and Steel Institute.

The autumn meeting of the Iron and Steel Institute will be held at Buxton from Sept. 26 to 30, 1910 (Monday to Friday).

On Monday, Sept. 26, there will be a reception, conversazione and concert, by invitation of the reception committee, the chairman of which is the president of the Institute, the Duke of Devonshire.

The first professional session will be held in the morning of Tuesday, Sept. 27, for the reading and discussion of papers. For the afternoon visits have been arranged to the works of the Midland Railway Company at Derby and to the Royal Crown Derby Porcelain Works.

On Wednesday, Sept. 28, a second professional session will be held in the morning, while in the afternoon visits will be paid to the Staveley Coal & Iron Company and the Broad Oaks Iron Works in Chesterfield. In the evening there will be a dinner at the Palace Hotel, in Buxton.

On Thursday, Sept. 29, an excursion will be made to the Crewe Works of the London & North Western Railway Company, and an alternative excursion to Manchester to visit the university, the Town Hall and Ryland's Library.

On Friday, Sept. 30, the Duke of Devonshire will invite the members and their ladies to a garden party at Chatsworth.

Mr. G. C. Lloyd, 28 Victoria Street, London, S. W., is the secretary of the Iron and Steel Institute.

The Western Metallurgical Field.

The Knudsen Furnace.

The report that the Knudsen furnace for copper matting is to be given a trial in Arizona has directed attention to a process of smelting that is not widely known in this country. Originally developed at the Sulitelma mines and reduction works in Norway, it has been in operation there for about five years, in units handling charges from 8 to 20 tons. The furnace is practically a basic lined converter into which a charge of fuel is dumped, followed by the charge to be smelted. A blast of air is turned on and the operation continued until a high-grade matte is formed.

The Sulitelma ore is a copper-iron sulphide, carrying about 15 per cent silica. It is mixed with silicious ore so that the charge smelted contains about 6 per cent copper, 36 per cent sulphur, 33 per cent iron, 4 per cent alumina and 21 per cent silica. In operating the furnace a charge of about 200 lb. of bituminous coal is charged and ignited, and a gentle blast turned on. The ore charge is then introduced and the blast continued at low pressure until the charge is molten, after which a full pressure of about 20 lb. is applied. Apparently any grade of matte can be produced according to the time allowed, and blister copper could probably be made. When the matte has been smelted to the desired grade the furnace is discharged into a settler from which the matte is tapped. It is stated that slags of low copper content, say, 0.5 per cent, are readily secured. The success of the furnace at Sulitelma will encourage a fair trail of the furnace in this country, especially in those districts where pyritic smelting would be an advantage.

Sale of the Clark Copper Interests.

The sale of the copper interests of Senator W. A. Clark to the Anaconda Copper Mining Company will give the latter practical control of the high-grade copper properties in the Butte district. This will result in the closing of the Butte Reduction Works as a copper producer. The 500 or 600 tons of ore formerly treated daily at this plant will be diverted to the Washoe plant at Anaconda. On account of the large capacity of the latter, this quantity of ore can be handled without difficulty in addition to the regular daily shipments from the Anaconda Company's mines.

By a slow process of elimination, due to one cause or another, Butte has gradually declined as a copper smelting center, although maintaining an eminent position as a copper mining district, and supplying ore to the two large reduction works at Anaconda and Great Falls. With the closing of the Butte Reduction Works there will be but one smelting plant left at Butte and the city will be practically free from the disagreeable effects of smelter smoke.

Basic-Lined Converters.

In the minds of some metallurgists the basic-lined Bessemer converter for copper matte is destined to supersede the one now generally used. These devices have been contemplated for some time and have been tried in several places. The Omaha plant of the A. S. & R. Company installed one about two years ago, but it is reported to have given poor satisfaction. Better reports, however, come from the Garfield Copper Company, Utah, where the basic-lined converter is in operation and giving sufficient satisfaction to warrant the erection of three more. Magnesite bricks of special shape are used in the lining and the life of the converter is sufficiently long to make the cost of converting compare favorably with the same operation in acid-lined converters. As in many other metallurgical operations, local conditions must play a large part in the advantage or disadvantage of this process.

Other suggested improvements in the bessemerizing process deal with the shape of the converter and the manner of applying the air blast. The suggestion of a "vortex" spherical converter is referred to at length in the Synopsis of Chemical and Metallurgical Literature in this issue. Mr. Haas, the inventor of this idea, also favors a basic-lined converter, although he does not advocate the use of magnesite brick.

Zinc.

The demand for zinc ores, which was rather brisk a short time ago, is falling off. A number of causes are contributing to this condition. Foremost is the plentiful supply, and in this factor the Butte district is exerting a strong influence. Apparently the future of Butte as a zinc producer is limited only by the desire of the operators to fill the demands made upon them. It is significant that in the transfer of the Clark interests to the Anaconda Company, Senator Clark retained the use of the zinc concentrator of the Butte Reduction Works, where the concentration of zinc ore is being carried on successfully. It is understood that the exploration and development work which has been done by Mr. Clark in the past few years will be continued until the zinc business becomes an important element in the future of Butte.

The smelting capacity of the zinc reduction works of the United States is not on the increase, and in fact there are a number of idle plants. In 1909 there were 30 plants in existence, with 211 furnaces and 90,550 retorts. Several of these plants are now idle and the number of retorts is reduced by at least 10,000.

Smelter Fume Disposal.

The different methods being adopted by various companies in disposing of smelter fumes without discharging into the atmosphere substances deleterious to vegetation are worthy of note. One of the first companies to observe the mandate of the law was the Ducktown Sulphur, Copper & Iron Company, of Isabella, Tenn. The chamber process of making sulphuric acid was installed there, and about 150 tons of acid are now made daily. Details of this plant and process are referred to in the Synopsis in this issue. The system of condensing the fumes by effecting a combination with zinc oxide or calcium oxide has been installed at the U. S. Smelting Company, at Midvale, Utah. This was reviewed in our April, 1910, issue (page 203). At the plant of the Selby Smelting & Lead Company, Selby, Cal., the Cottrell process of electric deposition is used, and the same process is being installed at the Balaklala smelter, Coram, Cal. Finally, at the Mammoth Copper Company's plant, at Kennet, Cal., there is being erected a structural steel baghouse for the condensation of furnace fume. This structure will contain about 3000 woolen bags, each 30 ft. long and 18 in. in diameter.

Recent history seems to indicate that the careless discharge of smelter fume into the atmosphere is rapidly becoming unpopular, and that the future will see but little or none of this practice. Different systems of disposal or utilization are available for plants operating under varying conditions, and undoubtedly one of these will be found adaptable at different plants.

Cripple Creek District.

The reported decision of the Colorado Springs & Cripple Creek District Railroad to advance rates of ore transportation from the Cripple Creek mines to the cyanide mills at Colorado Springs, has caused the Golden Cycle Company to build an aerial tramway from its mines to the mill. The equipment will cost about $200,000 and the plant will be in operation before the contract with the railroad company expires in April, 1911. This will place the Golden Cycle Company in an independent position, as it already owns mines and one of the best cyanide plants in the State.

Colorado School of Mines.

The thirty-sixth annual commencement exercises of the Colorado School of Mines was held in Golden, May 27, 1910. Pursuant to the custom, which has been followed for many years, of having an address by someone prominent in the mining or metallurgical world, the speaker on this occasion was

Mr. A. A. Blow, mining engineer and a former trustee of the school.

The subject of Mr. Blow's address was "The Responsibility of a Mining Engineer." In the course of his remarks, Mr. Blow stated that one of the desirable qualities of a mining engineer is a rational and persistent optimism, which will give him self-confidence and enable him to discharge his duty to his employer with a full appreciation of his responsibility in the matter.

Mr. Blow regarded the experience of personal responsibility as the most valuable experience an engineer could acquire, and urged the students to accept such responsibility whenever it offered itself.

The Stoiber prizes, which are awarded annually to the students presenting the most meritorious theses in metallurgy, were received by R. M. Keeney and G. M. Lee, whose work was on the direct manufacture in the electric furnace of steels and ferro-alloys; and Jean McCallum and E. J. Bruderlin, who submitted a design for a dry chlorination plant.

Utah Society of Engineers.

At the May meeting of this society the following officers were installed for the ensuing year: President, O. A. Hannold, electrical engineer, Utah Light & Railway Company; first vice-president, C. H. Repath, chief engineer, International Smelting & Refining Company; second vice-president, F. E. Arnold, designing engineer, The J. J. Burke Company; secretary, W. C. Ebaugh, professor of chemistry, University of Utah; treasurer, B. F. Tibby, consulting engineer.

The Non=Ferrous Metal Market.

For some weeks past the non-ferrous metal market has been practically without feature.

Prices have not ranged widely, and such fluctuations as have been produced have been without material effect one way or the other.

Consumers generally seem to be following a hand-to-mouth policy, and to be satisfied with it, and it has been noticeable that most of the purchases have been for immediate consumption.

The impression prevails that manufacturers are not plentifully supplied with stocks, but they seem content to pursue their present policy.

Copper.—Lake has been steady and quiet at 12⅞ to 13c., only a moderate amount of business being transacted. With plenty of the metal for sale at these figures, and lower in some instances, there is no anxiety among the consumers. Reports have been current to the effect that the future production from the low-grade copper properties is to be effected at a low cost, and this, together with the fact that the world's visible supply of the metal is undoubtedly large, has contributed to a quiet market.

Lead.—The demand for this metal has been fairly good, and the market has been firm at 4.30 to 4.35c., New York, and 4.17 to 4.20c., St. Louis. Missouri lead has shown a specially strong demand.

Zinc.—The market has shown a tendency to decline under the slow demand for the metal that has been offered. Manufacturers are buying only for immediate needs. Prices have been hovering around 5c., St. Louis, with the customary advance for New York.

Tin.—The demand has been confined to the immediate needs of consumers and the market has not been marked by any very large sales.

The price has been fairly steady at about 33c.

Other Metals.—Aluminium is the only metal for which the demand continues strong. The market has been steady at 23¾ to 24c., and business is reported good. Prices for antimony are nominal in a dull market, at 7½ to 8½c. for various brands.

The Iron and Steel Market.

There was no material curtailment in pig-iron production in June, and the rate toward the close of the month was probably in excess of 28,000,000 tons annually, representing a decrease of but slightly more than 10 per cent from the maximum rate, which was 31,600,000 tons in February. The production in the twelvemonth ending with June may be estimated at 29,750,000 ± 150,000 tons, production in the second half of the twelvemonth probably exceeding that in the first half by about 200,000 tons. Three calendar years, 1906, 1907 and 1909, have shown outputs between 25,000,000 and 26,000,000 tons, but no calendar year has yet shown 26,000,000 tons, and no twelve-month ending with June has hitherto shown as large a production as the 26,202,985 tons made in the twelvemonth ended June 30, 1907. Despite the fact that a slight further curtailment in output is likely in July and August, the prospect for a large increase in tonnage thereafter is such as to make chances substantially even whether the second half production will exceed or fall short of 15,000,000, tons, and as that was approximately the output in the first half, the prospects are for approximately 30,000,000 tons in the calendar year.

The persistence of a really very heavy production, which is certainly going into consumption, is remarkable, considering the entire reversal in sentiment which has occurred during the half year. Liberal forward buying and the accumulation of stocks by jobbers and manufacturing consumers have given way since late last year to curtailment of stocks and hand-to-mouth buying. Since the curtailment in production has been only about 10 per cent, it seems certain that the actual rate of ultimate consumption has been greater in the past two months than it was six months earlier.

The continued sagging tendency in prices and the general depressed tone of the market have been due not to a lack of tonnage considered absolutely or by comparison with the tonnages of previous years, but to its insufficiency as compared with capacity, which is represented, as to pig iron, by about 33,000,000 tons a year. Either the profits are still fairly large, in relation to actual capital investment, or there is a smaller spread between the best and worst positioned producers than would be assumed from a general survey of the industry.

During June the placing of new orders for finished steel products was very light, while specifications on old contracts were at a fairly satisfactory rate. Few branches of the finished steel trade reached July 1 with an accumulation of business sufficient to maintain even fairly full operations. As many contracts expired June 30, and are likely to be renewed quite leisurely, the volume of business in July promises to be very light.

A number of steel works with their attendant finishing departments are scheduled to stop July 1 for a fortnight or longer for inventory, repairs and accumulation of specifications. The iron mills of the Central West will have a general closing, their business being particularly light. The two labor organizations controlling most of the iron mills in the Central West, the Amalgamated Association and the United Sons of Vulcan, have demanded important wage advances for the year beginning July 1 and a positive agreement or disagreement on the scales will probably be deferred until August.

Pig Iron.

Minimum prices of pig iron at valley furnaces, 90 cents higher delivered Pittsburgh, are as follows: Bessemer, $15.50; basic, $14.50; No. 2 foundry, $14.35; forge, $13.60, representing declines during the month of 50 cents on Bessemer and basic, 65 cents on foundry and 90 cents on force. These lower prices are approximately $1 per ton below the cost of production to merchant furnaces which must buy their coke and ore, as most of them do, and this can be shown fairly conclusively by estimates of actual costs and by comparisons with previous low points, particularly in 1904 and 1909, when the striking of

higher prices than these, having regard to the difference in costs of wages, coke and ore, was followed promptly by advances. The situation is peculiar. There is a large profit in the Lake Superior ore, although some ore producers, working relatively expensive deposits, have much higher costs than those which made the region famous.

A few merchant furnace interests in the Central West have ore of their own, and can follow the pig iron market down still farther. The furnaces which must buy ore did not promptly withdraw from the market, having fairly large stocks in some cases, while apparently they were in doubt whether the market would ever react in their favor. While there is no prospect of any reduction in ore for this season, it would be absurd not to expect a moderate reduction for next season and it may not be long until this prospect will begin to influence a market which might otherwise advance.

After sales of Southern iron at $11.25, Birmingham, confidently reported in some quarters as amounting to 60,000 tons or more, the Southern market seems to be a trifle firmer, with prices held at $11.50, Birmingham, for prompt and third quarter, and $12 for fourth quarter. Buyers, however, are not quite ready to take hold at this range.

Steel.

Unfinished steel is slightly easier in tone and prices are quoted about 50 cents lower, as follows, f.o.b. maker's mill, Pittsburgh or Youngstown district: Bessemer billets, $25 to $25.50; sheet bars, $26 to $26.50; open-hearth billets, $27.50 to $28; sheet bars, $28.50 to $29; rods, $30.50 to $31.

Finished Products.

Plates and shapes have yielded fully a dollar a ton in price, and are quotable at not over $1.45, Pittsburgh, with occasional sales at $1.40. Indeed, in rare cases plates have been done at $1.37½. Sheets are slightly lower, and regular prices, as quoted below, are shaded about $2 a ton, except in the case of blue annealed, which are lower only by the disappearance of premiums over the regular price, for quick delivery. Steel bars have held up remarkably well.

Regular prices f.o.b. Pittsburgh are as follows:

Standard steel rails, $28 for Bessemer; $30 for open-hearth, f.o.b. mill, except Colorado.

Plates, $1.40 to $1.45 for tank quality, ¼-in. and heavier.

Shapes, $1.40 to $1.45 for I-beams and channels, 15-in. and under, zees and angles 2 x 3 and larger.

Steel bars, $1.45, base. Iron bars, $1.55, Pittsburgh; $1.45 to $1.50, Philadelphia; $1.45, Chicago.

Wire nails, $1.85, base; plain wire, $1.65, base; galvanized barbed wire, $2.15; these prices regularly shaded 5 cents per 100 lb.

Black sheets, 28-gage, $2.40; galvanized, $2.50; painted corrugated roofing, $1.70; galvanized, $3; blue annealed, 10-gage, $1.75. Tin plates, $3.60 for 100-lb. cokes.

Wakeman Luncheon.

A farewell luncheon was tendered at the Engineers' Club in New York to Mr. James M. Wakeman, retiring vice-president of the McGraw Publishing Company, on June 21, his birthday. On account of ill health Mr. Wakeman will shortly sail for Europe for a prolonged rest.

Some 130 friends of Mr. Wakeman were present. Mr. T. C. Martin was as graceful a toastmaster as ever. Speeches were made by Messrs. James H. McGraw, John A. Hill, Arthur Warren and Hugh M. Wilson. The McGraw Publishing Company presented to Mr. Wakeman a beautiful set of the works of his favorite author, Mark Twain, while the members of the advertising staffs presented him with a magnificent loving cup.

This journal is indebted to Mr. Wakeman for many a good advice, sincere criticism, and friendly encouragement, and in acknowledging this, joins his many friends in wishing him godspeed.

CORRESPONDENCE.

Titanium in the Steel Industry.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—I note in your June issue an abstract of a paper by C. E. Stromeyer, of Manchester, England, which purports to give some facts concerning titanium as an antidote for the brittleness of mild steel and its action in connection with phosphorus and other impurities. In the discussion of Mr. Stromeyer's paper, which followed in the Iron and Steel Institute, of London, his statements were torn to shreds by such well-known steel experts as E. H. Saniter, of Rotherham; Steel, Peech & Tozer, Ltd., of Sheffield; Hjalmar Braune, the celebated Swedish expert; Harbord, Twynam and others. Mr. Saniter, who is to receive the gold medal of the Institute this year for his efforts and researches in steel, stated among other things that "it would be noted in the investigation by all four of the above made by different men over a period of 30 years that the minimum of nitrogen found was 0.006 per cent. On referring to the nitrogen results in Mr. Stromeyer's paper it appears that the latter is again to be congratulated on the purity of his material, as 16 out of 20 steels (excluding the high phosphorus samples) showed less nitrogen than the minimum found by the other observers noted."

Now, lest we inadvertently do Mr. Stromeyer an injustice, let us reinstate his paper for a short discussion of its merits as relating to titanium and the elimination of nitrogen to some extent or to no extent as the facts may develop. Mr. Stromeyer makes the following remarkable statement: "It seems as if nitrogen when once it has entered the pig iron in the blast furnace cannot be removed by subsequent heating. The suitable chemical," he continues, "is believed to be titanium, but the addition of ferrotitanium alloy to steel has not demonstrated that this beneficial action takes place," and he goes on to show (sic) that he thinks titanium possibly does do the work of removing nitrogen, for in visiting the Leeds' Museum he found a specimen of titanium nitride.* Such indefatigable research scarcely requires an answer, so I quote another astonishing piece of information from this paper: "Of late, since the production of practically pure titanium has been affected, its addition to steel and cast iron has been attended with varying degrees of success, more particularly in removing blow holes."

On the strength (sic) of this glowing tribute to a new alloy he claims to have reduced the infinitesimal quantity of 0.0014 per cent of nitrogen down to the still smaller proportion of 0.0013 per cent by the addition of some "practically pure titanium." Now, this statement shows the futility of undertaking to answer a paper like Mr. Stromeyer's. Who can state that he had been able to determine the element of nitrogen in iron or steel down to 1-10,000 of 1 per cent? How can metallic titanium with its high melting point per se of, say, 5000° Fahr. affect molten iron or steel to any extent whatever?

To illustrate exactly what I mean to distinguish between the museum results, as quoted by Mr. Stromeyer, and the results actually obtained in the use of ferrotitanium in iron and steel, permit me to quote briefly from men whose names are not used except in connection with facts. Dr. H. Goldschmidt says in a paper read at the International Electric Congress, 1904: "Through the introduction of a small quantity of ferrotitanium which binds the nitrogen the composition of the bath undergoes a chemical improvement. Impurities, such as slag and others, are driven up and the bath is further purified. It is remarkable that the sulphur content is appreciably reduced."

E. von Maltitz, for years holding the position of metallurgical engineer with the Illinois Steel Company, says in *Stahl und Eisen*, No. 41, 1909: "In order to secure complete results it is necessary to employ not metallic titanium, but an alloy of

*Referring to the discovery inside the dismantled blast furnaces of the Farnley Iron Company of some crystals of titanium nitride, which are now in the Leeds Museum (p. 360 of June issue of this journal).

titanium and iron. By far the best results are obtained with an alloy containing 10 to 15 per cent titanium and no more. . . . The addition of this alloy is simple and meets with no difficulties."

P. Delville, in a paper read at the International Congress of Metallurgy, Liege, 1905, says: "The effects of titanium used as ferro in a cast-iron bath is to diminish considerably or to eliminate entirely the blow holes. This takes place by chemical action, reduction of the oxides dissolved, elimination of nitrogen," etc.

Dr. A. J. Rossi, the pioneer in titanium research and most indefatigable worker in its development, says: "I find that any steel or any iron which may have been improved and refined and strengthened with nickel or with chrome or with any other alloy may be still further purified and still further strengthened by the addition of a small quantity of ferrotitanium. Its limit as a strengthener in iron and steel has not yet been fully ascertained, but that it is the most powerful agent for strengthening and improving the baser metals is now beyond all cavil."

Dr. G. B. Waterhouse, the well-known metallurgist of the Lackawanna Steel Company, in *Iron Age*, June 2, 1910, says: "It has been his (Waterhouse) good fortune to be in the closest touch with the manufacture of over 140,000 tons of steel, in the making of which 1,310,000 lb. of ferrotitanium alloy has been used. The most suitable contents of titanium in this alloy have been found to be from 10 to 14 per cent. . . . Using an alloy with less than this percentage would not give any further advantage. As to the amount of nitrogen removed from the metal by the use of the alloy conclusive and definite experiments have not been made. One day the iron gave 0.015 per cent nitrogen and the finished steel 0.007 per cent; another day 0.021 per cent and 0.014 per cent respectively, and on still another day 0.015 per cent and 0.014 per cent. What evidence there is, therefore, points to improvement in this respect . . . and it has been found that the steel treated with the alloy shows much less segregation than the ordinary steel. Improvement has also been noticed in the working of the steel in the rolling mills and although the time has been short (from one to three years) which the rails have been in service on the different roads they have been found to give much better service than ordinary Bessemer steel rails."

From the foregoing it would seem evident that Mr. Stromeyer was somewhat at sea in his premises and in his conclusions. From the standpoint of the writer's experience it is safe to say that the story of the results to be secured by the use of titanium alloy will never be entirely complete until the day when iron and steel are no longer used as a means of carrying loads or stresses and of improving the strength to be secured by the use of substitutes for these metals.

I have only to say in conclusion that the largest steel makers in this country without an exception are using this alloy, and hundreds of the smaller manufacturers are using it as well.

Pittsburgh, Pa. CHARLES V. SLOCUM.

Furnace Electrodes.

To the Editor of Metallurgical and Chemical Engineering:

SIR:—The discussion of a paper by the writer on this subject, at the recent Pittsburgh convention of the American Electrochemical Society, showed that there still exists a doubt in the minds of some that the final and much simpler method devised by the writer for determining the proper proportions of electrodes for furnaces is still subject to the former criticisms concerning an assumption of constant conductivities. As these final results are free from all doubt based on such criticisms, and can therefore be safely used, the writer gives below the substance of his closing and unpublished remarks to that discussion, which include a brief demonstration of the final method based on those new quantities, so that anyone may satisfy himself as to its correctness and its independence of any assumptions concerning the conductivities; if so, then it is of general application. This non-mathematical demonstration is not supplementary to the others, but replaces them, hence the older ones need no longer be considered.

The general plan of this proof is to first assume certain new, definitely defined quantities, then to determine their values for any one particular electrode in accordance with those definitions, and then finally to show that these quantities are truly specific and therefore apply to other electrodes of other dimensions. This may not be an orthodox method of proving something, but I believe its correctness cannot be questioned, and it is certainly far simpler than any algebraic proof would be.

In this explanation the two conductivities are dropped out of consideration entirely as factors in the calculation. Hence, neither their values nor the temperature variations of these values need be known or considered; they may be great or small, and they may vary much or little—it does not matter; there are no conditions stipulated or involved. The only assumption is that these conductivities will always have the same fixed and definite values for the same material at the same temperatures—that is, that they are definite specific physical properties; no further assumption is necessary.

Instead of these conductivities, let us adopt two entirely new and different physical quantities, which we have selected and defined to be the watts per ampere, denoted by E, and the section per ampere and per inch length, denoted by S'; we, of course, have a right to make these new quantities anything we please. Their numerical values are then determined in any specific case in accordance with these adopted definitions, and must always agree with those definitions.

These two quantities really measure what we want to know about electrodes; unfortunately they have as yet no names; it is as though the ohm had not yet been named and was therefore called the "volts per ampere"; but the lack of a name would not mean that it is not a true physical property, namely, resistance.

Let it be assumed now (to be proved later) that for each range of temperature these two quantities are true specific properties—that is, that like specific gravity, specific heat, specific resistance, etc., their values are independent of the dimensions of any particular piece. And just as the numerical values of these other specific quantities change somewhat for different conditions of temperature, pressure, etc., so the numerical values of these new quantities change somewhat for different temperature drops in the electrode; hence, such values are given either in the form of tables, or (as in that Pittsburgh paper) in the form of equations.

Let it also be assumed that all electrodes are always operated so that there shall be no flow of heat through the hot end and that this, therefore, is the only condition of operation under consideration at any time; let this be referred to as operation "under normal conditions."

Being true specific quantities (so far, by assumption only) they may be determined experimentally once for all for any given material, with an electrode of any one selected size, and for each of any desired number of temperature ranges, and then tabulated as constants; just as the specific resistance may be determined from a piece of a metal of any selected size under each of a number of conditions of temperature or pressure or tension, etc. This is what I did, as described in the other Pittsburgh paper; the values of E and S' were therein calculated according to the above definitions.

Now, by definition, $E =$ watts $\div$ amperes, just as resistance might be defined as volts $\div$ amperes. Hence, as this is a definite specific quantity for each range of temperature, it follows without any further proof or mathematical deductions that the watts lost in any given electrode operating under normal conditions are equal to the value of E (for that given temperature drop) multiplied by the current. Hence, this method of calculating electrodes is proved as to E.

It will be seen that in this demonstration of the writer's method, the conductivities and their variations are completely eliminated from the calculation and from all consideration, although they and their variations are really what determine the result; they may vary with temperature or not, and little or much; their values and variations are all taken care of very effectively in the tabulated values of E.

In precisely the same way, by definition, $S' =$ section $\div$ (ampere $\times$ length). Hence, it follows (if this is a specific quantity) that the cross-section of any electrode operating under normal conditions, is equal to the value of S' (for that given temperature drop) multiplied by the current and the length. All that was said about determining the watts lost applies similarly to this section, and the result as to S' is therefore also free from any assumption as to the values of, or the constancy of, the conductivities, although they really determine that section.

I hope this is hereby made clear, and that the persistent claims of some academicians that my final method is based on constant conductivities and that it is, therefore, unreliable, will hereby be completely set at rest.

It remains to be demonstrated that these new quantities are true specific quantities under the normal electrode conditions. Although this will appear self-evident to anyone giving it due thought, the following proof based on the simple, neat and ingenious one, devised by Dr. E. F. Northrup, and referred to in my papers, ought to satisfy even the most skeptical academicians who have persistently doubted its correctness. It might be added here, for their benefit, that we are here concerned only with electrodes of uniform cross-section from end to end, and that the Thomson effect, heterogeneity of material, lateral leakage, chemical changes in the material, and all other like minor matters which so greatly complicate the complete mathematician's equations, are here considered to be nonexistent.

If it can be shown that the values of E and S' for any one particular range of temperature are the same for any cross-section and for any length then they are true specific quantities.

Imagine an electrode while operating under normal conditions to be cut in two, lengthwise. It is evident that this will change nothing in its operation, as neither heat nor current was traversing across that plane. Hence, one of these halves may be removed without causing any change in the other in which the current and the cross-section will then be just half of the original; the heat gradient of the remaining half will, of course, remain the same. The watts lost in the remaining half will, therefore, now be half of what they were, and will, therefore, still be equal to the same E multiplied by the current through that half, namely, by half the original current. The same will evidently be true for any other fractional part of the original cross-section, hence for all sections. That is, E is the same for all sections, and is, therefore, a true specific quantity in so far as the cross-section is concerned. No questions concerning the conductivities or their variations are here involved; they may have any values, great or small.

Now imagine this original electrode to be cut transversely into a very large number of extremely thin laminæ, like the leaves of a long row of books. Suppose now every alternate leaf was removed, and the rest brought together, making a new electrode just half as long. It is supposed, of course, that the contact resistance between the leaves is neglected or is always the same per leaf. Now, to produce the same amount of heat and the same temperature conditions in each one of the remaining laminæ as were before produced in two neighboring ones will evidently require twice the current that passed through it before; hence, the current must be doubled.

Nothing will hereby be changed in the heat gradient except that it will be condensed or shortened to half its length; the total drop from end to end is the same by definition; the value at the middle, or at 1/10 or 1/100 of its length from either end, will be the same in one as it is in the other, because the total

drop is the same. In other words, the heat gradient will be twice as steep at every point; the drop in each leaf will be the same as it was in two leaves before. The new heat gradient will be like the old one seen through a cylindrical lens, or drawn to a shorter horizontal scale. This is true no matter how irregular the shape of the curve of this heat gradient was, provided only that the laminæ were thin enough and numerous enough.

Now, the resistance of this new electrode will be just half as great, as one of each pair of minute laminæ has been removed. The current being doubled, and the resistance halved, the watts will be doubled; hence, they will be again equal to the same E multiplied by the new current. The same argument applies equally well to any other length, hence to all lengths. Therefore, E is the same for all lengths and is therefore a true specific quantity in so far as the length is concerned.

Having proved the same for the section, it follows that this quantity E is independent of both section and length and is, therefore, a true specific quantity.

By the same method it can be shown that the other quantity S' is also a true specific quantity. If, for instance, the original electrode is cut in two lengthwise, the value of S' is the same for each half because the current is half as great in each; hence, S' is the same for all sections. In the second case, if half the transverse laminæ are removed as before, we have found that the current will be twice as great, and, therefore, the value of S' will again be the same, because the length has been halved, the value of S' by definition being $S' =$ section $\div$ (ampere $\times$ length). Hence, S' will be the same for all lengths. Being the same for any cross-section and any length it is a true specific quantity. Here again we are not in any way concerned with the values of or the variations of the conductivities, they being taken care of completely in the experimentally determined values of S'.

In introducing the quantities k and r in that paper, the writer did so merely to show how their electrode mean value could be determined from these new quantities if it is desired to know them; but it was clearly stated that k and r were no longer involved as factors in the present simplified method, in which they need not be known or considered at all.

In view of the criticisms made by some academicians of the writer's original paper in the preceding volume of these Transactions, it may be of interest to add that the present simplified treatment of this subject, the correctness of which it is believed will no longer be questioned, gives exactly the same results as the original and more complicated method described therein is used—that is, when the constants are described therein is used, that is, when the constants are determined as therein described. Hence, if the present treatment is correct, the complete method described in that former paper is correct and does not involve the supposed errors on which so much stress was laid by some of the critics, none of whom could, however, show that there were any real errors, but simply assumed them to exist because their mathematical treatment became too complicated to give a solution. The case is of interest as showing how an approximate mathematical treatment (and the unfinished highly mathematical solution offered by one of the critics in the last volume of the Transactions is at best only approximate) can be extremely complicated, when the mathematically exact one (as I believe the present one to be) is ridiculously simple.

Philadelphia, Pa.   CARL HERING.

The International Electrotechnical Commission has just issued its first annual report. So far the work of the commission seems to have been chiefly in effecting organization. At present the following countries are represented in the commission: Belgium, Brazil, Canada, Denmark, France, Germany, Great Britain, Hungary, Italy, Mexico, Spain, Sweden, United States. Prof. Elihu Thomson is the president of the commission.

A Quantitative Investigation of the Source of Hydrogen in Blast Furnace Gas.

BY R. J. WYSOR AND W. D. BROWN.

The dearth of available literature concerned with the subject of this paper induced the authors to undertake the present investigation. Doubtless the most classical memoir on the influence of hydrogen in the blast furnace has been furnished by Sir I. Lowthian Bell in his "Principles of the Manufacture of Iron and Steel." So far as we are aware, no attempt has previously been made to account quantitatively for the presence of hydrogen in blast-furnace gas.

The general impression, sometimes voiced in text-books, that most of the hydrogen in the gas is due to dissociation of the aqueous vapor in the air blast was proved to be erroneous, by calculation from the available analyses which we have examined. In the expectation that all the excess hydrogen, over that produced from the water vapor in the blast, originated from the coke charged into the furnace, this investigation was designed to indicate the amount of hydrogen resulting from each source. Although the outcome of our experiments has not substantiated the original idea, we believe the results here presented have an intrinsic value in furthering present knowledge of the subject, and in encouraging a new scientific thought as to the ultimate source of hydrogen in blast-furnace gas.

At the outset, it was realized that it would be difficult to obtain ideal conditions of experimentation, such as could be observed or measured with strict accuracy. All due precautions, however, were taken to maintain these conditions under close control. All of the operations of the test, subsequently described, were personally performed by the authors or under their direct supervision.

Number 5 furnace at the plant of the Duquesne Steel Works & Blast Furnaces was selected as the basis of the test. Its chief dimensions, height and bosh diameter, are 94 ft. and 23 ft. respectively. This furnace had been in blast for about one year, and had been blown out for repairs to the lining a short time previous to the test. The air blast was delivered in part by a cross-compound, vertical, steeple-type, steam-blowing engine; the remainder by a twin tandem, horizontal, gas-blowing engine.

A preliminary experiment of seven hours' duration was followed by a 24-hour test, the results of which appear in this paper, as it was deemed necessary to have a continuous record for a considerable length of time in order to apprehend general working conditions.

Details of the Test.

The coke used came from one mine in the Connellsville district. Large, representative samples were secured at two-hour intervals from the chutes of the furnace bins, as the coke was being emptied into the buckets. The initial and final samples were taken about two hours before the beginning and end, respectively, of the continuous test. The remaining burden of the furnace, chiefly ore and limestone, had been previously sampled and analyzed. The pig iron produced was represented by samples regularly secured for analysis.

Owing to a constant leakage of condensed steam into the intake valves of the compression tubes of the steam blowing engine, it was impracticable to effect moisture determinations in the air blast by the use of a psychrometer in the immediate vicinity. These determinations were made in the air blast passing through a 6-in. pipe connected to the blast main near the hot-blast stoves; a strong current of air was maintained through this line. A small petcock, situated on the 6-in. pipe, afforded an easy means for the control of pressure. The air current was forced through a suitable absorption apparatus containing concentrated sulphuric acid, a meter completing the train. Continuous samples of approximately 12 cu. ft. each were taken over two-hour periods for the entire 24 hours. The relatively low moisture results obtained in the air blast are due to the

fact that the average temperature record of the test, conducted in April, was less than 60° Fahr.

Continuous gas samples were obtained in large aspirator bottles, at a uniform rate, over periods of time corresponding to those of the moisture samples. The samples were secured from the gas main leading from the dust catcher and were analyzed immediately.

Furnace Data.

The furnace was in good condition and worked with fair uniformity over the whole period. No tuyeres were removed on account of water leakage, nor could the furnace men detect any such leakage. Several small slips occurred, none of them exceeding 8 ft., and none occasioning loss of stock through the explosion doors. The blast pressure averaged about 14 lb., the calculated, approximate volume of air delivered to the furnace being 39,950 cu. ft. per minute. The hot-blast temperature fluctuated between the limits of 900° Fahr. and 1350° Fahr., the average being about 1150° Fahr. The average temperature of the top gas was approximately 550° Fahr. The ratio of coke to pig iron produced was 1.11, and that of the limestone 0.51.

Analytical Determinations.

All due precautions were taken in the various analyses, the results of which appear in subsequent tables. An average coke sample representing the entire time of the test was prepared for the ultimate analysis. Published statements as to the loss of hydrogen in coke samples due to fine grinding are not decisive. For the hydrogen determinations in this work, the samples were rather coarsely pulverized and the estimations were made upon the undried coke, due correction being effected for the moisture content.

Some attempt was made to determine the manner in which the sulphur existed in the coke, but aside from the fact that no appreciable amount was present as sulphate, the analytical evidence was not conclusive. However, in the absence of sulphur existing as sulphate, it may be shown by calculation that whether the sulphur is present in the organic form, as iron pyrites, or as ferrous sulphide, a negative correction of three-sevenths the iron content of the coke should be applied to the apparent ash to obtain the true ash. It happens in this instance that three-sevenths of the iron content equals three-eighths of the sulphur, the value generally accepted for effecting the correction, the iron (0.76 per cent) being present in just sufficient quantity to combine with all the sulphur as iron pyrites.

Although it is impracticable to utilize the individual analyses in the calculations, they are presented in considerable detail to indicate the range of variation.

Analyses.

The tabulated results of the moisture determinations in the air blast, all due corrections having been made, are calculated on the basis of dry air at standard conditions. The corresponding moisture determinations in the atmosphere, made by means of a psychrometer, are also presented. The excess moisture in the air blast is due to the steam leak into the compression tubes,

	MOISTURE.			GAS.				COKE:
	Grains per Cu. Ft.		Carbon Di-oxide.	Carbon Mon-oxide.	Hydro-gen.	Me-thane.	Nitro-gen.	Ash.
	Atmos-phere.	Air blast.						
1.	2.91	3.36	14.3	24.8	4.0	0.0	56.9%	11.08%
2.	2.39	3.31	13.4	25.4	4.0	0.2	57.0	11.08
3.	2.39	3.49	13.1	26.4	4.5	0.1	55.9	11.48
4.	2.52	3.53	12.4	26.8	4.5	0.0	56.3	10.86
5.	....	3.59	13.1	24.0	5.3	0.1	57.5	11.27
6.	2.66	3.67	12.8	25.9	5.5	0.1	55.7	13.22
7.	....	3.55	14.0	24.3	4.2	0.1	57.4	13.30
8.	2.40	3.07	13.6	24.6	4.3	0.2	57.3	12.32
9.	2.55	2.97	13.4	25.3	4.0	0.1	57.2	12.48
10.	....	3.00	12.9	25.0	3.9	0.2	58.0	11.24
11.	2.70	3.17	13.7	24.0	3.8	0.1	58.4	12.51
12.	....	3.23	13.2	24.4	3.6	0.1	58.7	13.92
Average Gas Analysis	...13.3		25.1	4.3	0.1	57.2		

	Carbon.	Hydro-gen.	Oxy-gen.	Nitro-gen.	Ash.	Sul-phur.
Average Coke Analysis	86.12	0.55	0.44	0.35	12.00	0.86

Average Moisture in Air Blast..................... 3.33 grains per cu. ft.
Average Carbon in Limestone Charged............ 11.1 per cent.
Average Carbon in Pig Iron Produced............ 4.0 per cent.

previously mentioned. The numbers designate the sequence of samples taken over two-hour periods, or at two-hour intervals. The percentages of ash in the coke samples are expressed to indicate somewhat the variation in composition. Hydrogen estimations in several samples showed the percentage of this constituent to be fairly uniform.

Calculations.

It is hoped that the subsequent methods of calculation by volume may be explained with sufficient clearness as to be readily intelligible to those not at present familiar with this system.

Hydrogen Derived from the Coke.

The percentage of hydrogen in the gas originating in the coke may be determined from three factors: the hydrogen and carbon entering the gas, and the percentage of carbon compounds in the gas. The ratio by weight of hydrogen to carbon gasefied is converted to the ratio by volume of the hydrogen to the carbon compounds in the gas by multiplying by six; this factor is deduced by the following reasoning.

One molecule of hydrogen contains two atoms. One molecule of carbon monoxide, carbon dioxide or methane contains one atom of carbon, having a weight equal to 12 atoms of hydrogen. According to Avogadro's law, the molecules of all gases occupy equal volumes. Therefore, two atoms of hydrogen, or one molecule, occupy a volume equal to that occupied by a molecule of the gaseous carbon compounds, containing one atom of carbon, which weighs 12 times as much as the atom of hydrogen and six times as much as the hydrogen molecule. Conversely, for unit weights of hydrogen and carbon entering the gas, the ratio of volumes of hydrogen to carbon compounds must be as 6 to 1. The calculations are tabulated as follows:

Hydrogen in coke.............................. 0.55 per cent
One-eighth oxygen in coke (0.44%)............. 0.055 per cent
Residual hydrogen in coke..................... 0.495 per cent
Coke used per ton of pig iron................. 1.11 tons

Residual hydrogen per ton of pig iron......................... .00549 ton

Carbon in coke................................86.12 per cent
Carbon used per ton of pig iron............... 1.11 tons
Carbon in coke per ton of pig iron............................ .9559 ton
Carbon in limestone..........................11.10 per cent
Limestone used per ton of pig iron........... 0.51 ton

Carbon in limestone per ton of pig iron.....................0.0566 ton

Total carbon entering the furnace per ton of pig iron........1.0125 ton
Carbon in pig iron............................ 4.0 per cent
Carbon in a ton of pig iron................................. .04 ton

Carbon gasefied... .9725 ton

Ratio by weight of hydrogen to carbon gasefied....... $\dfrac{0.00549}{0.9725}$ = 0.00565

Multiplying above ratio by 6, we obtain:
Ratio by volume of hydrogen to compounds of carbon in gas.. 0.0339
Sum of percentages of carbon dioxide, carbon monoxide and
    methane in gas.. 38.5

Hydrogen in gas.. 1.30%

Hydrogen Derived from the Moisture in the Air Blast.

Moisture in the air blast on coming in contact with incandescent coke reacts with the formation of one molecule of hydrogen from one molecule of moisture. Moisture in the air, being unsaturated, acts as a perfect gas and, in accordance with Avogadro's law, occupies a volume equal to that of the hydrogen derived from it. Therefore, the actual volume of hydrogen in the gas is equal to the volume of moisture in the air blast. However, for statement in percentage, correction must be made for the ratio of gas to air volume. This correction is based on the percentage of nitrogen in dry air and in the gas. By

Moisture accompanying 1 cu. ft. dry air @ 62°F. and 30"
    mercury .... 3.33 grains
Weight of 1 cu. ft. aqueous vapor @ 62° F. and 30"
    mercury .... 332.03 grains
Volume of 3.33 grains @ 62° F. and 30" mercury........ 0.0100 cu. ft.
Hydrogen entering gas from 1 cu. ft. dry air.......... 0.0100 cu. ft.
Nitrogen in gas.................................... 57.2%
Nitrogen in air................................... 79.1%
Ratio of gas to air blown............................. 1.38

Hydrogen derived from moisture per cu ft. gas.. $\dfrac{1.38}{1.38}$ = 0.0072 cu. ft.

Percentage of hydrogen derived from moisture........... 0.72%
Hydrogen found in gas by analysis .... 4.3 %
Hidrogen in gas calculated from coke and moisture in air
    blast 2.02%

calculation the nitrogen in the gas derived from the coke is 0.07 per cent of the gas, and is omitted from the calculations as being negligible.

Conclusions.

The result of this test demonstrates that some other source or sources than the two discussed in this paper were responsible to a large extent for the hydrogen in the furnace gas. The most obvious inference would be that the extraneous source was from a leakage in the cooling-water system around the furnace, or in the water-cooled valves at the hot-blast stoves. We do not believe an appreciable error could have existed at the furnace, as the furnace men claim the ability to detect a very slight leakage. It was impossible to determine definitely whether or not a leak existed in one of the water-cooled valves though they were examined and represented to be in good order; no trouble was detected from this source until a considerable time after the test.

By calculation it may be shown that with the conditions obtaining in this test there must be a leakage of approximately 3.1 gal. of water per minute to yield 1 per cent of hydrogen in the gas, or nearly 7 gal. per minute would be required to produce the hydrogen unaccounted for in the gas. We believe a leakage of no such magnitude could have existed throughout this test.

Likewise it may be proved by calculation, or by simple examination of the foregoing figures, that one grain of moisture per cubic foot of air blown is responsible for only 0.22 per cent of hydrogen in the resulting gas.

We desire to call attention to the fact that numerous analyses made in this laboratory, of gas samples caught directly after slips of considerable magnitude, show an increased percentage of hydrogen, sometimes reaching a very high figure. One sample obtained directly after a 21-ft. slip showed the abnormal result of 39 per cent hydrogen. It is difficult to account for such high percentages of hydrogen except by the theory that the water, present as moisture or in the combined form, carried in the burden is dissociated when the stock drops down into contact with incandescent coke. Of course, the hydrogen in the fresh coke additions to the furnace might also be expected to be liberated with greater rapidity.

Further, might not this dissociation occur at a higher level in the furnace than is generally supposed? The Mesaba ores used in the furnace contained between 5 and 6 per cent combined water. It is known that a fairly high temperature is required to expel the water of hydration of certain silicates. We believe it is not impossible, nor even improbable that part of the combined water suffers dissociation with the consequent liberation of hydrogen. This point has not been tested experimentally.

In consideration of the rapid progress being made in the installation of large gas power engines, using blast furnace gas as fuel, and the deleterious influence of high percentages of hydrogen in the explosive mixtures used, the question of hydrogen in the gas, its quantity, and its source, is one of peculiar interest.

Duquesne Steel Works, Duquesne, Pa.

Convention of American Institute of Electrical Engineers.—The annual convention of the Institute will be held from June 28 (Tuesday) to July 1 (Friday), at Hotel Waumbeck, Jefferson, N. H. The program contains 20 papers, dealing with lighting, high-tension transmission, telegraphy and telephony, industrial power and traction matters. Among the entertainment features are a reception and dance on Tuesday evening at the Waumbeck and a trip to Mount Washington, which will take practically the whole of Thursday. For Thursday evening a dinner and discussion by the sections committee and section delegates has been arranged; for Friday night, a dinner, followed by a discussion by the educational committee. The professional program contains also the presidential address of Mr. Lewis B. Stillwell and a report of the standards committee, which offers certain standardization rules for approval.

Chloridizing Roasting of Cuprous Pyrites Cinders.

By Sten Lilja.

In the middle of last century iron pyrites began to be commercially used in the manufacture of sulphuric acid. From the beginning, it was thought that the residues from the roasting operation, on account of their content of such valuable elements as copper and iron, ought to be further treated, and a number of prominent metallurgists have bent their efforts toward developing methods by which the copper could be extracted and the residues, or cinder, converted into a commercial product.

To extract the copper by means of chloridizing roasting, followed by leaching, was early proposed by Orshall, Longmaid, Maumené, Bechi and Haupt, Schaffner, not to mention others. Longmaid obtained a patent dated Oct. 20, 1842, and another in January, 1844, both relating to the treatment of cinder by roasting it with NaCl, but his real intention seems to have been to produce principally salt cake and chlorine.

Bechi and Haupt were operating, in 1856, a plant located in Capanne Vecchi, where they were recovering copper by a similar process.

Henderson carried the process to a greater perfection. In 1865 he built a factory at Hebburn (Bede Metall. Comp.) with the object of extracting copper from pyrites-burners by a method which he had protected by patent. It was Henderson who introduced the absorption towers through which the acid gases from the chloridizing roasting operation have to pass.

The Henderson plant at Hebburn was from the start a great success to such a degree that in five years, according to Wedding, not less than 20 plants were operating in a similar way, treating together 250,000 tons of cinder yearly. Since then the Henderson method, or the chloridizing roasting process, is exclusively used for extraction of copper from pyrite cinders, by which process the residues incidentally are converted in a form more suitable for the ultimate recovery of the iron.

The chemical principles of the chloridizing roasting, as given by Henderson 45 years ago, still stand as correct, but the mechanical arrangements naturally have developed with the progress of modern time, lowering the cost of treatment and thus making the process more economical.

The greatest number of plants using this method and now operating are situated in England, in the Tyne district and in South Lancashire. The others are distributed over Germany (Duisburg, Königshütte), Belgium (Hemixen), Sweden (Helsingborg), United States (at the Pennsylvania Salt Manufacturing Company's factories in Philadelphia and Natrona), Austria (Wittkowitz, near Mährisch-Ostrau).

Henderson Process.

The principal features of the Henderson process can be described shortly as follows:

Cuprous cinder is mixed with a certain proportion of common salt (the amount depending upon the sulphur and the copper content) and the mixture crushed to 8 to 10 mesh. After crushing, it is subjected to chloridizing roasting in a reverberatory or a muffle furnace at as low a temperature as possible (about 1100° Fahr. being most advantageous) The following reaction will take place:

$$CuS + 2NaCl + O_4 = Na_2SO_4 + CuCl_2.$$
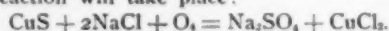
CuCl₂ is soluble in water. The aim is, therefore, to convert the total copper in the residues into this form, but on account of the chemical and physical conditions of the residues and the imperfect method of operation, other reactions will take place, causing a resulting product with only part of the copper soluble in water, but with all the rest soluble in weak acids. To facilitate the extraction of copper at the leaching, the acid gases —HCl and SO₂—which develop during the roasting, are absorbed in towers, and with the diluted acid solutions thus obtained the roasted residues are treated after having been transferred to leaching tanks.

In the cupriferous lyes the silver is precipitated with KI and afterward the copper with scrap iron. The silver precipitate contains about 6 per cent of metallic silver, which can be recovered by smelting. The copper precipitate—usually called cement copper—contains 50 per cent to 95 per cent copper. It is directly smelted and refined.

The final residue, after the leaching, goes under the trade name of purple ore or blue billy. On account of its high iron content it is a valuable raw material for blast furnaces, but must on account of its fineness first be briquetted.

Nearly all the cuprous pyrites, i.e., iron pyrites with over 1 per cent Cu, which is being used in the manufacture of acid, is produced in Portugal, Spain, Norway and Italy. Spain and Portugal produced in 1907 about 3,500,000 tons of cuprous pyrites. The chief producers in Spain are the Rio Tinto Works and Tharsis Sulphur & Copper Company.

The Rio Tinto Works' production of cuprous pyrites was, according to *Mineral Industry,* in long tons.

	Locally treated.	For shipment.	Total.	Per cent Cu.
1907....	1,265,090	641,858	1,906,948	2.417
1908....	1,115,610	604,275	1,719,885	2.265
1909....	1,184,188	604,795	1,788,987	2.349

The copper in the locally treated pyrites is extracted by smelting or by the famous Rio Tinto leaching process (described in the *Mineral Industry,* Vol. XII, p. 112; XV, p. 288; XVII, p. 278).

The exported pyrites are sent all over Europe and to the United States.

Norway exported in 1908 not less than 250,000 tons of pyrites. The two biggest producers were Sulitjelma and Foldalen, the former with a production for the year of nearly 100,000 tons, while the latter shipped about 50,000 tons.

The greater part of the Sulitjelma ore is desulphurized at sulphuric acid plants and sulphite mills in Norway and Sweden, and the Cu in the residues extracted partly at Helsingborg, Sweden, and partly at Hemixen, Belgium. The copper content in the Norwegian pyrites varies from 2 per cent to 5 per cent.

In the United States, according to *Mineral Industry,* 206,471 tons of pyrites for acid making were produced in 1908. This pyrites contains generally little copper.

The pyrites imported, 1909, according to one of the biggest import firms, was 669,992 tons, with following Cu content:

232,028@1.80 per cent Cu.

281,196@0.73 per cent Cu.

156,773 under 0.25 per cent Cu.

According to the burning the best results with highest copper recovery are obtained by chloridizing and leaching when the pyrites are uniformly roasted, not being sintered and not containing kernels of green ore. There ought to be still 1 to 1½ times as much sulphur in them as the amount of copper they contain.

By the burning, the pyrites will lose in the average 30 per cent of their weight, so that the copper contents in the residues will be correspondingly increased (10/7 of the original).

For Rio Tinto ore it can be seen that with 2.35 per cent Cu in the crude pyrites, the cinder will contain 3.34 per cent in average. This Cu content varies considerably in shipments from time to time. For instance, during last year the Rio Tinto pyrites which was imported in the United States did not contain more than 1.6 per cent of Cu, making the copper content in the residues 2.30 per cent.

Arrangement of Plant.

The most advantageous location of a chloridizing plant for the treatment of pyrites residues is, of course, in close proximity to the acid manufacturing plant, so that there will be a minimum of transportation. In some cases it might be necessary to locate the plant at some central point to where raw material from several acid works can be brought together. A large plant treating a great tonnage can be operated more economically than a smaller one. It is then also possible to arrange for further treatment of the purple ore in the same place.

On the arrival of the residues at the works, they have to be carefully stored, preferably under a roof, as too great a percentage of moisture will cause not only trouble in the crushing and screening, but also added cost for the evaporation of the water in the roasters and a lowered production. Another reason for great care in the storing of the residues is the variation of the sulphur content in material from different desulphurizing plants. The proper way is to test incoming cinder for sulphur before unloading and place it in the storage sheds according to sulphur content so that in each compartment there is always cinder of a certain grade. It is then easy in the operation of the chloridizing roasters to mix the residues so that the charge always contains a uniform amount of sulphur. This facilitates the work and insures good results.

In describing more in detail the chloridizing roasting, we can to advantage subdivide the operation into the following five processes:

    I. Mixing with salt and the crushing of the mixture.
    II. Roasting proper.
    III. Leaching.
    IV. Precipitation of silver.
    V. Precipitation of copper.

I. Mixing with Salt and Crushing.

The mixing with salt, with the subsequent crushing, must be carefully done for a successful roasting and leaching. It has been the custom to spread the residues in a layer on a platform or the floor and add 12 per cent to 15 per cent of salt, and if needs be, some green ore. The whole mass was afterward thrown with shovels directly into the grinding machine. In some places there is employed a mechanical mixer consisting of a rotary drum, where the ground cinder is mixed with salt. When the grinding and mixing take place simultaneously the simplest way is to feed them to the grinder by means of some feeding apparatus, by which the proportions can be easily governed. This will incidentally make the feed to the grinder more uniform, which is beneficial for an even grinding. The crushing and grinding machines which are usually found in plants of this kind are Cornish rolls, ball mills, Chilian mills, Carlin mills and Kent mills. The last mentioned are best suited for fines. When used for lump cinder, a preliminary crusher is required.

The fineness to which the material is ground depends to some degree on the nature of the residues, but usually they are comminuted to 8 to 12 mesh.

The necessity for a uniform grinding can be understood from the fact that too large grains would at the chloridizing be only partially chloridized. The material can be allowed in the kiln only a certain time, as otherwise the fines would be "dead roasted." A too fine grinding, on the other hand, would at the subsequent leaching cause too much slime that makes trouble at leaching.

After the crushing, the material is preferably stored in bins, from where it is drawn at intervals to be charged into the kilns.

II. Roasting.

Kiln Construction.—The first type of kilns used for chloridizing roasting was the reverberatory kiln with the flame passing directly over the ore. On account of the uneven temperature caused in the kiln by the direct flame, the results were usually unsatisfactory, because, in the part of the charge which was nearest the fireplace, the copper, too much heated, was transformed into oxide—"dead roasted."

To overcome this disadvantage an arch was built across the hearth nearest the fireplace over which the flame would strike; in this way a kind of muffle would be formed, open toward one side. This construction was used at the Bede Metall. Comp. in England.

Another type of reverberatory furnace is used at Widness, in Lancashire. It is fired with producer gas with the combustion gases passing in channels under the hearth in the length direction of the kiln and returning over the ore to the stack flue.

All the reverberatory furnaces have the common drawback that the outgoing roasting gases are mixed with the combustion gases, increasing the difficulties of the absorption of the acids causing a direct loss as the acids for the leaching have to be acquired from elsewhere.

At nearly all the chloridizing roasting plants the ore is, therefore, treated in muffle kilns. Of the several types of such kilns that have been designed and are in use, the most common type is the one that is employed, for instance, at Tharsis Sulphur & Copper Company's works. It is described in Lunge's "Sulphuric Acid and Alkali." The hearth is 9 ft. to 14 ft. in width and 30 ft. to 40 ft. long, and takes a charge of 2.5 tons to 4.5 tons. The combustion gases pass through channels around the muffle and do not come in direct contact with the ore, which, therefore, is more uniformly heated. The draft of the combustion and of the roasting gases are regulated independently of each other. A similar type of kiln has been used by the Pennsylvania Salt Manufacturing Company.

Figs. 1, 2 and 3, showing a muffle furnace for chloridizing roasting, are reproduced from Schnabel's "Metallurgy."

Another type of furnace was used for several years at Pitkaranta, in Finland, and some features of this construction were
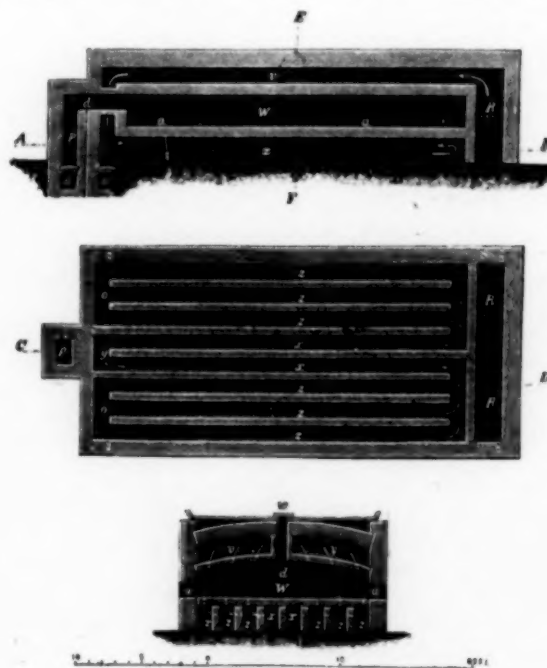

FIGS. 1, 2 AND 3.—MUFFLE FURNACE FOR CHLORIDIZING ROASTING.

adopted, and are still used to some extent, at the Helsingborg plant. The Pitkaranta furnace consists of four muffles 6 ft. in width, 1 ft. high and 12 ft. long placed over each other and connected by vertical holes, through which the ore is raked from muffle to muffle in the same way as in a malétra furnace. The muffles are open at both ends to facilitate the operations. For each furnace there are four fireplaces, located two and two under the lowest muffle. The combustion gases pass around the muffles and escape through two horizontal conduits running over the furnaces, which are arranged together. The gases from the ore are collected by other conduits lying parallel with those just mentioned.

Mechanical Roasters.—The chloridizing roasting in hand-operated furnaces is rather expensive and more or less dependent on the skill of the workmen. It has, therefore, been the aim to substitute the hand labor by some mechanical device. As early as 1872 Gibb patented a mechanical roasting furnace, in detail described in *Dingler's Polytechn. Journal* No. 204, page 288, and in Lunge's "Sulphuric Acid and Alkali," Part I.

The kiln consisted of a round hearth, 15 ft. in diameter, supported in the center by a vertical shaft rotating in a step bearing. The hearth was enclosed by brick walls and a brick arch. The fireplace was at the base of the furnace and the combustion gases were allowed to pass over the ore to the flue of the stack opposite the fireplace. By an endless chain running in a groove around the hearth and driven by a tooth wheel on a vertical transmission axle outside the furnace, the hearth was rotated. To distribute the ore, which was fed centrally, there was used a system of obliquely arranged rabble plates, suspended from above and together reaching from the center to the periphery of the hearth. The plates were lifted out as soon as the material was evenly distributed. During the rotation of the hearth the ore was continuously stirred by a plow-shaped raker, which, by some mechanical contrivance, was pushed back and forth over the hearth. Figs. 4 and 5, showing Gibbs' mechanical furnace for chloridizing roasting, are reproduced from Schnabel's "Metallurgy."

When the roasting was finished, the rabble plates were again lowered and the ore was automatically discharged at the continued rotation of the hearth. The usual charge was 5 tons, for
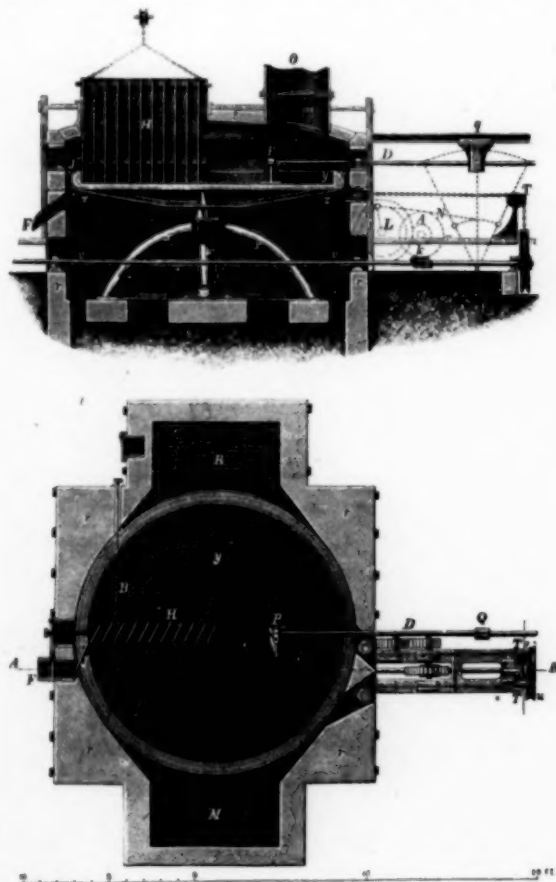
FIGS. 4 AND 5—GIBBS MECHANICAL FURNACE FOR CHLORIDIZING
ROASTING.

the roasting of which nine hours were required. Gibb states that the ore was better roasted than at the hand-operated kilns. More copper was soluble in water and less salt was consumed. It was used for some years at the Bede Metall. Works, but is now superseded by later constructions. A drawback in this construction was the same as at reverberatory furnaces, i.e., combustion gases and roasting gases were mixed.

Later, mechanical roasting kilns of various types have been tried at several places. The Pearce Turret-kiln, for instance, is used at Natrona, Pa. At the Falun Copper Works, in Sweden, the White Howell kiln was used for some time. The results were not satisfactory, and the use of the kiln was discontinued.

The same principle as in this kiln was found in the type employed at the Eastern Nodulizing Works, at Hackensack River, Newark, where a cement kiln was adapted for chloridizing roasting. It is said that the results were good. This type of kiln would seem, however, not to be quite suitable for this purpose, partly for the reason that the flame comes into direct contact with the ore when the roasting is complete, whereby $CuCl_2$ is decomposed, partly on account of the mixing of the roasting and combustion gases as in the reverberatory furnaces.

The Wedge Furnace (this journal, Vol. VI, p. 457, VII. p. 225) is a distinct improvement on all the previous types. When used for chloridizing roasting, one hearth is employed of 32 ft. diameter. From a vertical rotating axle in the center which supports horizontal with rabble plates furnished arms, the ore is stirred and moved forward from the center to the discharge at the periphery. The combustion gases pass through muffles over or under the hearth.

For the recovery of the acid gases developed in the chloridizing roasting process, they are led through absorption towers built in the same way as those employed in the manufacture of muriatic acid. These are filled with coke, clay pipes or something similar.

The gas enters at the bottom and meets water dripping down from the top, thus the most part of the HCl, SO_3 and CuCl that the roasting gases contains is obtained in a weak acid solution 2-5° BB, which is led in wooden pipes to the leaching department.

Operation.—According to Wedding, the method of operation at the use of reverberatory furnaces is as follows: The ore after the charging is spread out and is slowly heated up until it becomes red hot nearest the fireplace. The ore then is turned over and the firing stopped. It is then left lying for a few hours without firing, but with excess of air, during which time the temperature decreases to dark-red color. Now a vigorous stirring of the ore with broad iron bars and hand rabbles follows, which will cause a raising of the temperature to a light cherry red.

In the beginning a greater part of the gases develop and blue flames indicate that sulphur is oxidized. By and by the flames decrease in strength until they finally disappear at the same time as the ore gets colder. Most of the sulphur is now oxidized. A sample is taken and a simple test is done to ascertain how soluble the copper is. If the test is satisfactory, the ore is discharged on the floor to cool off before leaching.

The usual test to try whether the roasting is completed is made in the following manner by the foreman or the roaster: Five grains of the material is boiled with weak (2 per cent) HCl for five minutes. The copper is then washed out with hot water.

The rest, which has not gone into solution, is boiled with aqua regia until the iron is dissolved. Ammoniac is added and the precipitate is filtered and diluted to a certain volume. The intensity of the color of the copper solution indicates whether the roasting is finished or not.

At the Helsingborg Copper Works the roasting in the Pitkaranta furnace is thus carried on. The crushed cinder is stored in bins, from which it is at intervals drawn into buckets traveling on a tramway to the conical bunkers over the roasting kilns. The bunkers are closed at the bottom by an iron ball, at the lifting of which a charge of 2000 lb. drops into the uppermost muffle. It lies here three hours, and is then raked down to the next muffle. The temperature has slowly increased, and at the end of the three hours it is so high that the reaction has begun, as can be noticed from the blue flames of burning sulphur.

On the second muffle the ore is distributed and left for a second period of three hours, at the end of which it is transferred to the third muffle. The sulphur flames are not any longer so pronounced and at the transfer of the ore to the fourth shelf, which is the lowest, most of the sulphur is eliminated.

After resting here three hours, the ore is finally discharged. While on the third and fourth shelf it is to advantage to occasionally stir the ore. The discharge opening in the lowest shelf is always left open for the entrance of the air required for the roasting, and all the time a uniform fire is kept in the fireplace.

To obtain an even roasting of the ore during the 12 hours it remains in the kiln it is essential that the S content always is the same. In some cases it is necessary to add some green pyrites. In other cases, when the S-content of the charge is too high, the feed is lowered, so that the same result is obtained as if the ore had been lying in the kiln a longer time.

It is of importance that the roasted ore is allowed to cool off without too much disturbance. At the discharge, the various salts are still in melted condition. At the cooling, if left alone, they will solidify in a porous mass easy to leach. At the Helsingborg plant the ore is discharged on inclined cooling plates located in the basement under the kilns. From there the

FIG. 6.—ARRANGEMENT OF THE TANKS AT EASTERN NODULIZING WORKS PLANT.

ore will slide into small cars running on an industrial railway. The cars are raised to the level of the leaching room by an elevator.

III. Leaching of the Roasted Cinder.

A convenient arrangement of the leaching department is to have the leaching tanks on a higher elevation than the precipitation tanks, so that the copper-carrying lyes can run by gravity from the former tanks to the latter ones. If the elevation of the ground is unsuitable, the leaching tanks can, for instance, be placed on an upper floor. After the treatment, the blue billy can be loaded directly into small cars which run out on a stock trestle to be dumped.

Fig. 6 shows how the tanks are placed at the Eastern Nodulizing Works plant, while Fig. 7 gives a general view of the leaching department at this plant.

The leaching tanks usually are built of wood, square or round, and are large enough for a 10-ton to 25-ton charge. They are lined with lead or bricks in cement or asphalt, which latter must be protected by an inside lining of wood.

A very good construction of leaching tanks is used at Helsingborg (Fig. 8), and is well worth describing in detail. The tanks are elliptical, with diameters of 10 ft. and 6 ft. and an inside depth of 4 ft. For building material Swedish resinous pine is used. The bottom and the staves are composed of 5-in. wood. The latter are held together by three iron bands 2 in. x 3¼ in. kept at a short distance from the sides of the tanks by pieces of wood so that lye leaking over the edges will not disintegrate them.

As near the bottom as possible there is a hole through the wall for the removal of the lyes. To the sides of the hole a lead pipe is attached. This is continued by means of a rubber hose with squeeze for the drainage of the tank.

Wooden launders have to be used for the transportation of the lyes to and from the tanks. For whatever pumping is required either steam injectors or air-lift pumps are employed.

In preparing the tanks for the ore standard bricks are placed on edge right on the bottom. On these, special perforated bricks, 12 in. square, are laid as a false bottom for a filter composed of straw or screened lump cinder, iron ore, or something similar, on top of which the ore is charged and leveled.

The leaching itself is done approximately in the same way at various places. At Helsingborg, for instance, after the leveling of the charge, acid lyes of 5° to 8° B strength, obtained in previous leaching, are let into the tank. The ore is still warm from the roasting. The first lye that is drawn is very strong (40° B). It contains about 5 per cent copper and nearly all the silver. It is, therefore, transferred immediately through a special launder to the silver precipitation department.

When the strength of the lyes as they are drawn has gone down to 22° B, they proceed to the copper precipitation. This continues until the strength of the outgoing lyes has gone down to 10° B. The incoming acid lyes are shut off and in their place acid from the towers is let into the tank. This acid is allowed to remain in the tank two hours, at the end of which time the lye is exchanged for fresh acid, which in its turn remains two hours. The lye so obtained is used as the first lye for freshly filled tanks.

After this leaching, the ore is washed with water, first with such as has been used at the washing of the previous tanks and finally with fresh water. To test whether all the copper is extracted, a well-polished iron plate is stuck down into the offgoing lyes. If there is no coloring of copper, the leaching is finished.

The wash water ought to have a temperature of about 50° C. The time of leaching is usually 40 hours for a 10-ton charge.

Of the copper in the roasted ore about

80 per cent is soluble in water,

16 per cent is soluble in weak HCl,

4 per cent requires HNO_3 to be put in solution, and will remain in the purple ore.

IV. Precipitation of Silver.

At the chloridizing roasting AgS will be converted into Ag Cl, which at the leaching is transferred into solution by the


FIG. 7.—GENERAL VIEW OF THE LEACHING DEPARTMENT AT EASTERN NODULIZING WORKS PLANT.

NaCl and $CuCl_2$ lyes. If the cement copper is to be refined electrolytically a special precipitation of the silver is not necessary. If this is not the case, it is usual to extract the silver before the copper precipitation.

Claudet's process is most common. At this the lyes, which contain 0.0025 per cent to 0.01 per cent silver, are precipitated with ZnI_2 or NaI.

The following reaction takes place:

$$2 AgCl + NaI = NaCl + AgI.$$
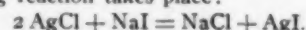
AgI forms a heavy insoluble precipitate, which is allowed to sink to the bottom in tanks prepared for this purpose. The precipitate, containing besides AgI also $PbSO_4$ and $CaSO_4$, is washed until it is clean from copper. It is thereafter treated with Zn and H_2SO_4, which causes the iodine to go into solution as ZnI_2 to be washed off and re-used.

Instead of this process with Zn and H$_2$SO$_4$, sometimes Na$_2$S is employed, the reaction then being 2AgI + Na$_2$S = Ag$_2$S + 2 NaI.

The NaI is washed out and re-used.

The first lyes from the leaching of the roasted cinder, which contain most of the silver, are transported to special tanks in which slime, mechanically carried over by the lyes, will settle together with some CaSo$_4$ and PbSO$_4$, which are precipitated at the cooling of the solutions. The remaining lye is analyzed for silver, a sufficient quantity of KI in a solution of 0.1 per cent to 0.2 per cent strength is added, and the mixture is well stirred, preferably with some arrangement operated by compressed air or mechanical means. For that reason it is better to carry out the precipitation in a special tank from which the lyes are transferred to other tanks constructed similarly to the leaching tanks, where the precipitate will settle.

After two or three days the remaining lye is drawn off. It may then still contain 2 mg. to 3 mg. Ag per liter. For emptying the tanks, they are furnished with two holes, one very near the bottom and the other about 8 in. higher. The desilverized lye is drawn through the upper hole, so that the precipitate in the bottom is left undisturbed.

New lye from the precipitation tanks is let in and the procedure is repeated until there is so much silver slime in the bottom that it has to be removed. This has usually to be done every month or two, depending on the original silver percentage.

The silver slime is drawn through the lower hole. Some gypsum, which has crystallized on the sides of the tank, is removed separately and treated. It contains 0.5 per cent silver. The slime is washed with hot water and some H$_2$SO$_4$ for dissolving copper salts, and is afterward treated with Na$_2$S or Zn and H$_2$SO$_4$ in rotating wooden cylinders. As soon as the iodine is completely washed out, the slime is taken to filter presses or drying kilns. It is then ready for melting. It contains 3 per cent to 10 per cent silver and usually some gold.
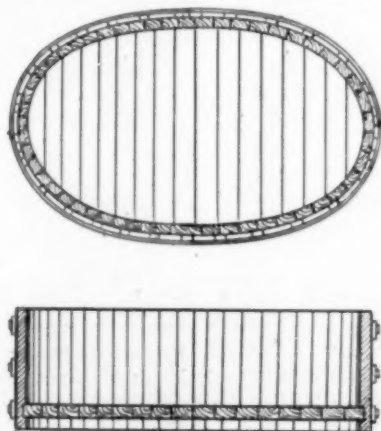

FIG. 8.—LEACHING TANK USED AT HELSINGBORG'S COPPER WORKS.

V. Precipitation of Copper.

For the precipitation of copper of the lyes from the leaching plant and the silver precipitation tanks, iron scrap is employed.

The tanks here also are built in the same way as the leaching tanks. Fig. 9 shows the precipitation tanks at the Eastern Nodulizing Works plant.

The tanks are filled loosely with thin scrap iron before the lyes are let in. These are heated with steam to nearly boiling temperature, which hastens the precipitation. The reaction is quickest when the iron is clean and the lyes are strong. They are left there until a test with a polished iron plate does not show indication of copper.

They are then exchanged for new lyes and this is repeated until the precipitation takes place so slowly that it is better to take away the precipitated copper and clean the scrap iron. It is on account of the copper covering all the iron that the reaction is delayed. The cement copper is removed with copper shovels and the iron pieces are collected on a copper sieve, while the fine copper is washed off with water.

The cement copper is further washed with hot water in special tanks for the removal of chlorides, and is then dried in drying chambers or by pressing out the water in hydraulic presses.

The decopperized lyes are transported through a series of safety tanks in which traces of copper, which they still may contain, are caught, together with fine copper slime, which is mechanically carried over. The finished lyes usually contain small percentages of Zn, Ni, Co, together with Fe and Na$_2$SO$_4$. It has long been desired to find some method by which these

FIG. 9.—PRECIPITATION TANKS AT EASTERN NODULIZING WORKS.

substances could be economically saved, but so far the problem has not been satisfactorily solved.

Treatment of Final Residue.

In the leaching the cinder is freed from the greatest part of its copper and sulphur content and incidentally from a number of other elements, such as Ni, Co, Mn, and the residues—purple ore or blue billy—consist chiefly of Fe$_2$O$_3$ with some SiO$_2$ and Al$_2$O$_3$, the qualities of which depend on the purity of the original pyrites.

In the blue billy from Rio Tinto pyrites, the iron content usually is 64 per cent. It can, therefore, be considered as a valuable iron ore, although its fine granulation makes its direct use in the blast-furnace charge rather disadvantageous. A large part of it is liable to be carried away as flue dust with the gases.

Nowadays it is, therefore, usually converted into lump form by some sintering process before it is used in the blast furnace. The nodulizing method and the Gröndal briquetting process are among the best known and the most successful processes. In the former, the cinder is passed through a long rotary cylinder similar to cement kilns, in which it is heated to a high temperature by means of powdered coal blown in at the discharge end. The product varies in size from that of rice to that of walnuts, and forms hard balls more or less thoroughly sintered.

By the Gröndal method the cinder is formed into briquettes of uniform size by an automatic plunger press. The briquettes are loaded on flat cars, which are slowly pushed through a channel kiln 150 ft. to 200 ft. long. Here they are gradually heated until they arrive at a zone of the kiln where the temperature attains the high degree of 2400° Fahr. As fuel, producer gas is usually employed.

The finished product is hard and strong, but porous, so that it is well suited to blast-furnace work and open-hearth practice.

Edward C. Hegeler.—By the death of Mr. Edward C. Hegeler, at Lasalle, Ill., on June 4, the zinc industry of this country has lost one of its most distinguished pioneers and founders. Together with F. W. Matthiessen, he founded the Matthiessen & Hegeler Zinc Co., at Lasalle, Ill., this place being selected as the most satisfactory contact point between the Wisconsin zinc mines and the Illinois coal field. The works started in 1860 and have been very successful ever since. Many important improvements in the zinc smeltry are due to Mr. Hegeler's originality, as the very long gas-fired furnace, labor-saving devices, and especially his muffle roasting furnace, introduced in 1881, which started the manufacture of sulphuric acid as a byproduct.

Furnace Electrode Losses

In our issue for May, page 238, we reviewed and abstracted Mr. Carl Hering's paper on furnace electrodes read at the recent Charlotte convention of the American Institute of Electrical Engineers. In the opinion of Mr. Hering we did not, in that abstract, give sufficient prominence to the fact that Mr. Hering now claims that his experimental results supply electric furnace designers with two quantities, E and S', which are independent of any consideration whatever of electrical and thermal conductivities of the electrode material and, for any given material, are functions simply of the temperature drop through the electrode, and are true whenever the electrode tip is at the furnace temperature, so that no heat passes through the furnace tip in either direction.

Mr. Hering obtains these quantities directly from his experimental results. He measures the amperes and volts passing when temperature equilibrium has been reached, the maximum and minimum temperatures, and the dimensions of the electrode. E is then the measured voltage from hot to cold end for this definite temperature drop through the electrode; while S' is the cross-section divided by the amperes flowing and the length for the same conditions of temperature equilibrium at the electrode tip in the furnace. As these values were based only upon experimental measurements, they are altogether independent of any assumptions concerning thermal or electrical conductivity of the electrodes.

Specifications for Foundry Coke *

By Dr. R. Moldenke.

The subject of specifications for foundry coke has been long agitated, but very little has so far been accomplished, and this probably because we have been going through just such a change in practice in our coke as we have in our pig irons ever since more liberal and yet more exact views have prevailed. The appearance of a coke no longer carries the weight it formerly did, and all that is now asked is that it melts iron properly and has a composition to do this without serious detriment to the metal.

This change in practice has undoubtedly been the result of the educational efforts of the "by-products" coke people, and as it has meant the saving of much money formerly spent in freight, the foundry industry has been the gainer in every direction on the coke item.

There is still much to be learned by the foundryman regarding the fuel he uses for melting purposes. Too often the blame for poor results is placed on the coke, when the real trouble lies in his melting practice. So that there may be no doubt about the fuel end, it is highly advisable that a good and yet liberal set of specifications for foundry coke be drawn up so that coke bought under them may assure the foundryman that on this item of his daily work he may be reasonably safe. If then he runs into trouble, he will have eliminated the coke from his study of the difficulty.

There are many points which have a bearing on the value of a coke and yet cannot well be added to such specifications. Thus, the cellular structure is an important factor on the working of the coke in the cupola. It must be remembered that in contradistinction to blast-furnace practice, where there is a reducing atmosphere, in the cupola it is oxidizing. In the blast furnace the cellular structure of the coke should be as great as possible, so that every molecule of oxygen is used up in burning the carbon, and with an excess of incandescent carbon make carbonic oxide gas. This means that the lighter the coke—consistent with its carrying power—and the more porous, the better.

On the other hand, for foundry purposes, it is desirable that coke be burned with a slight excess of oxygen, as in boiler

*A paper presented at the Detroit convention of the American Foundrymen's Association.

practice; so that as complete a combustion of the fuel as possible may result. That is, the gas should be carbonic acid with but little carbonic oxide. Now, this can best be accomplished by presenting smaller surfaces to the passing oxygen. For that reason the smooth surfaces of anthracite, with no cellular structure whatever, give a most excellent fuel for the cupola, though coke can be burned faster.

Considering average coke to be 50 per cent cellular structure, the blast furnace can take care of higher percentages, while the cupola should have them lower. Very light cokes are therefore to be avoided, and rather the denser, heavier ones—provided the weight does not come from an excess of ash—are to be recommended.

On the question of sulphur, we have still something to learn. As between the volatile sulphur and that fixed in the ash of coke, we do not yet know positively why sometimes with the latter we get more into our castings and at other times proportionally less. We know that temperature and slag conditions have much to do with this, but until we are more certain, a differentiation of the sulphur content of a coke is not yet safe, the idea being that the dangerous portion of the sulphur only need be kept down.

Further, there is no doubt but that we will eventually have some means of reducing the sulphur in our molten iron before pouring, and then sulphur in coke will lose some of its terrors.

The following suggestions are presented for specifications for foundry coke, and the following acknowledgments are made. The limits in composition and the base analysis are suggested by Mr. A. W. Belden, coke expert of the U. S. Geological Survey. They are the result of his extended observation with cokes from all the fields of this country. The portion relating to the shatter test comes from the regular practice of the Detroit "Solvay Coke" plant, Mr. Warren S. Baluvelt having kindly sent the writer the specifications on this point.

The subject of premiums for extra good coke as against penalties for running below the mark has long been advocated by the writer as the only fair way of handling the subject. So long as the limit for rejection is placed at a point which will do no injustice to the producer, he will study to get the best results so that he not only gains financially, but also acquires a reputation for it. Penalizing without giving corresponding premiums for special excellence has always been held as against fundamental law, and where the base analysis has been settled upon, and safe physical tests added to the chemical, the results should be good for consumer and producer alike.

The accompanying suggested specifications are presented for discussion herewith, and it is hoped that some progress be made in this important subject.

Suggested Specifications for Foundry Coke.

Coke bought under these specifications should be massive, in large pieces, and as free as possible from black ends and cinder.

Sampling.—Each carload, or its equivalent, shall be considered as a unit, and sampled by taking from the exposed surface at least one piece for each ton, and so as to fairly represent the shipment. These samples, properly broken down and ground to the fineness of coarse sawdust, well mixed and dried before analysis, shall be used as a basis for the payment of the shipment. In case of disagreement between buyer and seller, an independent chemist, mutually agreed upon, shall be employed to sample and analyze the coke, the cost to be borne by the party at fault.

Base Analysis.—The following analysis, representing an average grade of foundry coke, capable of being made in any of the districts supplying foundries, shall be considered the *base*, premiums and penalties to be calculated thereon as determined by the analysis on an agreed base price.

Volatile Matter	1.00
Fixed Carbon	85.50
Ash	12.00
Sulphur	1.10

Moisture.—Payment shall be made on shipments on the basis of "dry coke." The weight received shall therefore be corrected by deducting the water contained. (Note.—Coke producers should add sufficient coke to their tonnage shipments to make up for the water included, as shown by their own determinations.)

Volatile Matter.—For every 0.50, or fraction thereof, above the 1.00 allowed, deduct .. cents from the price. Over 2.50 rejects the shipment, at the option of the purchaser.

Fixed Carbon.—For every 1.00 or fraction thereof, above 85.50 add, and for every 1.00 or fraction thereof below 85.50, deduct .. cents. Below 78.50 rejects the shipment at the option of the purchaser.

Ash.—For every 0.50 or fraction thereof below 12.00, add, and for every 0.50 or fraction thereof above 12.00, deduct .. cents from the price. Above 15.00 rejects the shipment at the option of the purchaser.

Sulphur.—For every 0.10 or fraction thereof below 1.10 add, and for every 0.10 or fraction thereof above, deduct .. cents from the price. Above 1.30 rejects the shipment at the option of the purchaser.

Shatter Test.—On arrival of the shipment, the coke shall be subjected to a shatter test, as described below. The percentage of fine coke thus determined, above 5 per cent of the coke, shall be deducted from the amount of coke to be paid for (after allowing for the water), and paid at breeze prices, previously agreed upon. Above 25 per cent fine coke rejects the shipment at the option of the purchaser. Fine coke shall be coke that passes through a wire screen with square holes 2 in. in the clear.

The apparatus for making the shatter test should be a box capable of holding at least 100 lb. coke, supported with the bottom 6 ft. above a cast-iron plate. The doors on the bottom of the box shall be so hinged and latched that they will swing freely away when opened, and will not impede the fall of the coke. Boards shall be put around the cast-iron plate so that no coke may be lost.

A sample of approximately 50 lb. is taken at random from the car, using a 1¼-in. tine fork, and placed in the box without attempt to arrange it therein. The entire material shall be dropped four times upon the cast-iron plate, the small material and the dust being returned with the large coke each time.

After the fourth drop the material is screened as above given, the screen to be in horizontal position, shaken once only, and no attempt made to put the small pieces through specially.

The coke remaining shall be weighed and the percentage of the fine coke determined.

If the sum of the weight indicate a loss of over 1 per cent, the test shall be rejected, and a new one made.

Rejection by reason of failure to pass the shatter test shall not take place until at least two check tests have been made.

Watchung, N. J.

The Chemical Import Trade of the United States of America.

BY WILLIAM POLLARD DIGBY.

Taking the 10 years ending June 30, 1908, the yearly value of imports of chemical materials has averaged $63,274,000, the maximum being $82,998,000 in 1907 and the minimum being $42,682,000 in 1899. The annual fluctuations for each fiscal year from 1899 to 1908 in the sum total of this trade are illustrated in Fig. 1.

The imports generally greatly exceed the exports. Even in 1908, the year of maximum exports, the imports are about thrice the exports. This, of course, indicates the fact that despite the great industrial strides in this sphere, America must remain for some years dependent upon Europe as a source of supply. The high degree of specialization in the German aniline dye industry on the one hand and the British alkali industry on the other hand are, of course, contributory causes. Another factor is the cheapness of ocean freights as compared with railroad transit, and it is thus apparent that while a factory in the Middle or Western States would be able to snap its fingers at European competition so far as its immediate vicinity is concerned, that factory might be faced with serious competition on the Eastern seaboard.

Examining in detail the sources of the chief materials imported during each of the five years ending in 1908, and again dividing the sources of supply among the six trade zones of Great Britain, British North America, remainder of British possessions, Germany, remaining European countries and South America, we find the state of affairs set out in the following tables:

Sulphate of Ammonia.—This material was not separately enumerated before the fiscal year ending June 30, 1907; about 94 per cent of the imports come from Great Britain, the reason lying in the extensive use of coal gas in that country. The figures are given below:

	1907	1908
United Kingdom	$1,532,127	$1,628,544
Germany	1,574	126
Rest of Europe	44,479	23,458
British North America	25,530	7,983
British Colonies	56,327	128,401
South America		
Rest of world		
Total	$1,660,037	$1,788,512
Average price per pound	$0.027	$0.026

Coal Tar Colors and Dyes.—Germany's world renowned predominance in the aniline dye industry is well exemplified by the volume of her trade with America. The value rose steadily until 1907, there being a decrease of about $500,000 in 1908.
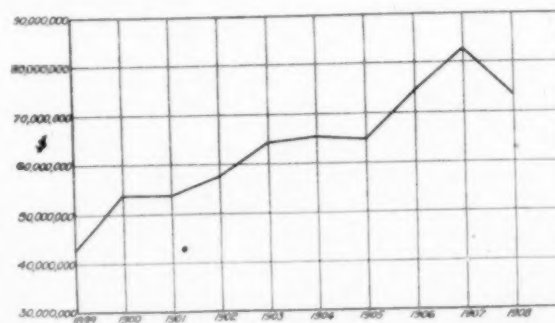

FIG. I.—TOTAL CHEMICAL IMPORT TRADE OF U. S. A. 1899-1908.

Perhaps now that the German chemical manufacturing companies have been compelled by recent legislation in regard to patents to open works in Great Britain, the values shipped thence to the United States may rise above the insignificant value here recorded:

	1904.	1905.	1906.	1907.	1908.
United Kingdom	$214,406	$196,393	$177,634	$138,202	$158,997
Germany	3,789,755	4,387,191	4,546,399	4,573,913	3,983,225
Rest of Europe	888,904	1,101,980	1,014,762	907,989	733,237
Brit. Nth. America	22,558	19,303	16,440	14,564	7,700
British Colonies		9	3	12	10
South America					
Rest of World	2,880	215	358	321	506
Total	$4,918,503	$5,705,091	$5,755,596	$5,635,001	$4,883,675

Chloride of Lime.—The electrolytic alkali industry is now the one undertaking of an electrochemical nature which is a commercial success in Great Britain; of course, a large proportion of the present exports are produced by the older chemical processes. Cheap raw material, and cheap coal, coupled with a home demand larger than that of any other country are factors tending for the success of this enterprise. Of course, given cheap salt and low railroad freights, there are important development of the electrolytic industry in the United States in conjunction with hydroelectric generating plants. The imports of this material are set out on p. 402. Note may be made in passing of the decline in 1908 which naturally accompanied the decline in the imports of aniline dyes.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$585,445	$579,385	$682,554	$710,020	$607,749
Germany	168,483	144,822	154,353	139,531	133,892
Rest of Europe...	14,098	51,621	41,407	30,813	12,859
Brit. Nth. America.	4,506	453	945		
British Colonies....					
South America....					
Rest of World....				1	
Total	$772,532	$776,281	$879,260	$880,364	$754,501
Average price per pound	$0.0097	$0.0081	$0.081	$0.0083	$0.0084

Carbonate of Potash.—In regard to all the potassium salts, Germany affords the chief source of supply. For this material and for **caustic potash** separate enumerations as to value were not made before 1907.

Carbonate of Potash.

United Kingdom	$9,098	$18,846
Germany	537,506	522,583
Rest of Europe	212,090	301,641
British North America........................	1,765	1,947
British Colonies		
South America		
Rest of world................................	286	119
Total	$760,745	$844,136
Average price per pound......................	.030	.034

Caustic Potash.

United Kingdom	$15,778	$19,603
Germany	159,312	177,565
Rest of Europe	91,769	55,713
British North America	186	
South America		
Rest of world................................		
Total	$267,045	$242,881
Average price per pound......................	.036	.040

Muriate, Nitrate and Sulphate of Potash.—Here again the trade is mainly German. The detailed sources of supply are set out below:

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$66,933	$65,068	$91,688	$98,765	$22,503
Germany	3,639,378	4,745,794	4,805,788	4,737,266	4,839,264
Rest of Europe....	272,220	265,242	252,327	9,791	24,528
Br. Nth. America.	8,333	2,733	3,179	2,376	372
British Colonies..	396,507	348,350	399,409	386,965	515,070
South America.....					209
Rest of world....	8,463	15,323	26,858	26,469	6,587
Total	$4,491,834	$5,442,510	$5,579,249	$5,261,632	$5,408,533

Nitrate cf Soda.—Pending the fulfillment of Sir Joseph Swan's forecast as to the coming prominence of this electrochemical industry, the United States in common with the rest of the world must look to the guano deposits of South
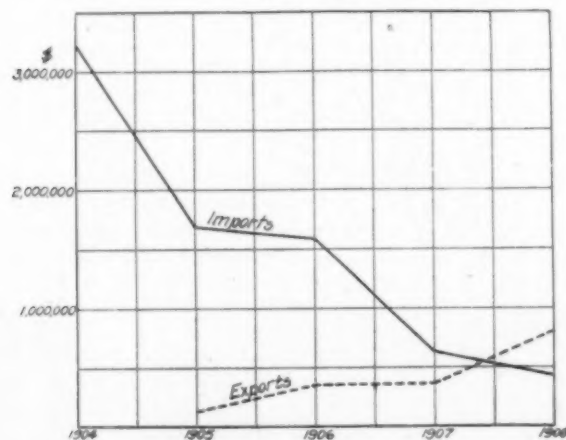

FIG. 2.—COMPARISON OF EXPORT AND IMPORT TRADE IN SULPHUR, U. S. A., 1904-1908.

America for her supplies. Figures are given below of the yearly value of this trade. The steady rise in price since 1904 is significant.

	1904.	1905.	1906.	1907.	1908.
United Kingdom...				$46	
Germany..........		2	66	2	1,354
Rest of Europe.					
Br. Nth. America.		5,882	2,096	848	230
Brit. Colonies.					
South America....	9,259,655	9,677,512	13,115,725	14,040,306	12,545,027
Rest of world.....	1				
Ttoal	9,259,656	9,683,396	13,117,887	14,041,202	12,546,611
Av. price per ton..	31.6	34.3	35.1	41.1	38.0

Caustic Soda, Soda Ash and Other Soda Compounds.—These alkaline substances whose production accompanies, and

whose use frequently precedes, chloride of lime, are furnished mainly by Great Britain. There are, however, marked signs of a diminution of this demand. Of course, part of the retrogression in the value of the imports in 1908 is attributable to the same causes as those which in this year checked the imports of aniline dyes and chloride of lime; and part to the development
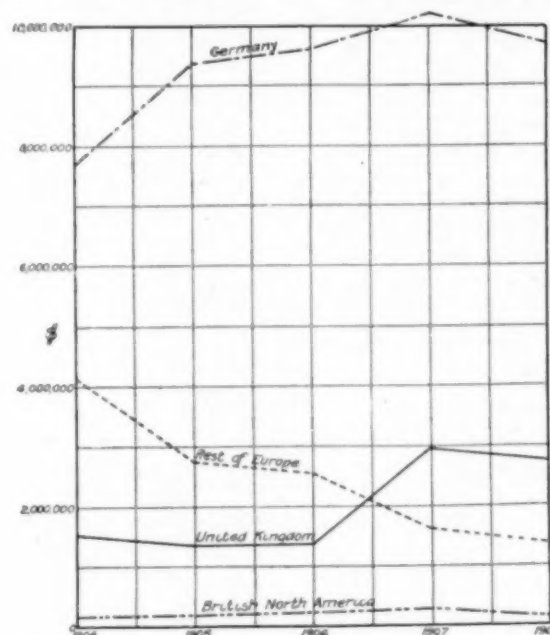

FIG. 3.—DISTRIBUTION OF CHEMICAL IMPORT TRADE, U. S. A., 1904-1908.

of the electrolytic caustic soda industry in the United States. The following table gives the relative values of the supplies furnished by different countries:

	1904.	1905.	1906.	1907.	1908.
United Kingdom.	$433,345	$380,968	$298,152	$320,838	$247,773
Germany	65,445	61,120	85,158	68,780	33,070
Rest of Europe....	62,951	25,047	30,263	20,315	8,240
Br. Nth. America.	219	2,372	754	30,511	159
Brit. Colonies......					3
South America.....					
Rest of world....	50	62	204	84	71
Total	562,010	469,569	414,531	440,528	289,316

Sulphur.—When reviewing the export trade in sulphur in our last issue we drew attention to the fivefold increase in exports in 1908 as compared with 1905. The figures of the import trade are equally startling. An import of $3,231,000 in value in 1904 decreased to $430,840 in 1908. Whereas, in 1904 supplies from Great Britain were worth $157,251, in 1908 they had fallen to $4,379, while the supplies from European countries other than Germany (of course mainly from Italy) fell in value from $2,855,000 to $211,000.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$157,251	$51,342	$55,503	$41,951	$4,379
Germany		1,531	308		40
Rest of Europe ...	2,854,827	1,305,605	1,216,359	305,512	211,144
Br. Nth. America..	60		472		485
British Colonies....	43				
South America.....		56	464		
Rest of world.....	219,442	327,663	324,457	290,759	214,792
Total	3,231,623	1,686,197	1,597,563	638,222	430,840
Av. price per ton..	19.1	18.4	18.1	18.1	16.4

Emery and Corundum.—Although considerable quantities of these materials are still imported, the very striking increase in the export trade for the last year under consideration has made itself felt in a decrease of the value of the amount imported, as the following figures show:

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$60,847	$72,576	$70,289	$76,072	$38,757
Germany	2,091	1,980	2,026	3,239	446
Rest of Europe....	53,662	27,988	13,951	10,421	4,587
Br. Nth. America.	41,183	90,998	147,210	151,239	79,831
British Colonies...	7,718	2,342	678	1,188	3,485
South America ...	2,118	1,824			
Rest of world.....	86,334	98,426	179,272	197,215	214,036
Total	253,953	296,134	413,426	439,394	341,142

Lime.—Accompanying the increase in the exports of this commodity to Canada, the imports have fallen more than 50 per cent in the last five years, although this country still accounts for about half the total trade. The distribution is shown in the following table:

	1904.	1905.	1906.	1907.	1908.
United Kingdom .	$1,329	$2,896	$7,146	$21,102	$18,902
Germany	10,003	12,479	14,074	14,665	13,422
Rest of Europe....	88	59		924	87
Br. Nth. America..	75,475	71,478	55,566	57,308	32,918
South America....					
Rest of world....	3,493	831	1,875	1,236	1,558
Total	90,388	87,743	78,671	95,245	66,890
Av. price per pound	.0017	.0019	.0019	.0024	.0032

A final table is added, showing the total distribution to the various trade zones, and is chiefly interesting in showing how the United States depends very largely on Germany and South America, on the former chiefly for aniline dyes and on the latter for manures. Great Britain being mostly responsible for general commodities.

	1904.	1905.	1906.	1907.	1908.
United Kingdom..	$1,519,656	$1,349,628	$1,382,966	$2,963,999	$2,766,053
Germany	7,675,155	9,354,919	9,608,162	10,235,788	9,704,987
Rest of Europe...	4,146,550	2,777,542	2,569,069	1,634,013	1,375,494
Br. Nth. America..	152,334	193,219	226,662	284,347	131,626
British Colonies...	408,776	341,154	401,035	444,492	646,970
South America....	9,261,773	9,679,392	13,116,190	14,040,306	12,545,239
Rest of world.....	320,663	432,520	533,025	516,084	437,547

Grüner's Ideal Working of a Blast Furnace.*

By Prof. Joseph W. Richards.

For many years the ideal perfection of blast-furnace running has been that promulgated by Grüner. This dictum states that the blast furnace works most economically when the largest proportion of the carbon of the fuel is oxidized by the blast before the tuyeres, and the smallest part oxidized above the tuyeres by the oxygen of the solid charges (reductions).

This ideal is true for most blast furnaces working under usual conditions; we hope to show that it is not true for some blast furnaces under usual conditions, and not true for many blast furnaces under certain unusual conditions.

We will now discuss

(I) Why it is true for most blast furnaces under usual conditions.

(II) Why it is not true for some blast furnaces under usual conditions.

(III) Why it is not true for many blast furnaces under unusual conditions.

I.

The critical part of a blast furnace is the smelting zone. At that place, a high critical temperature must be maintained in order to smelt the charges, and the necessary heat quantity in calories must be furnished. This heat is furnished by the combustion of carbon to CO gas before the tuyeres and by the sensible heat in the blast. These two factors govern the amount of heat available, and its initial temperature. Only CO gas can be formed in the smelting zone; enough carbon must be thus burned to produce the desired smelting effect upon the iron and slag.

In most furnaces the amount of carbon which it is thus necessary to burn, to produce the smelting effect, will furnish far more CO gas than is necessary to reduce the iron oxides in the charge.

It follows from this statement that since more CO gas is necessarily produced in the smelting zone than is necessary to reduce the iron oxides any use of solid carbon as a reducing agent above the smelting zone is a clear waste of carbon. Having more than sufficient reducing agent in the CO gas formed in the smelting zone, this gas alone should be the reducing agent for the charges.

It follows that in all furnaces with the usual quantity of slag to be melted, and the usual temperature of blast, economy of fuel is reached in measure as Grüner's ideal is approached, and

*A paper read before the International Congress of Mining and Metallurgy in Düsseldorf, June, 1910.

that oxidation of carbon by the oxygen of the solid charges should be a minimum in order to attain maximum economy of fuel.

II.

In furnaces with pure fuel, or pure ores, or using little flux, or combinations of these conditions, the heat required for smelting purposes rapidly diminishes. It may very easily happen that by a combination of these circumstances so little heat is needed in the smelting zone (per unit of pig iron made) that the CO gas necessarily produced in developing this necessary heat just suffices to perform the reduction of iron oxides in the furnace. In this case, Grüner's ideal working is still consistent with maximum economy of fuel. We will call this the "equilibrium point."

But, if the heat necessary in the smelting zone requires so little carbon to be there burned to CO gas that the gas produced is not sufficient in amount to reduce the iron oxides of the charge, then Grüner's ideal no longer holds, and carbon can be oxidized by the solid charges with increased economy of fuel.

Such conditions frequently happen in charcoal furnaces working with rich ores, which, therefore, produce very little slag. In such cases, Grüner's ideal working would be wasteful, and greater economy is attained by departing from it as far as is possible.

[A consideration of the usual reactions in the blast furnace shows clearly the economy of using up carbon in direct reduction if it is possible to dispense with burning it all at the tuyeres. The usual reactions are

$$9C + 9O = 9CO \text{ (at the tuyeres)}$$
$$Fe^2O^3 + 9CO = 2Fe + 3CO^2 + 6CO$$
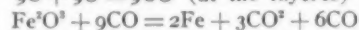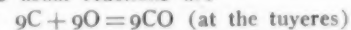
The limiting condition for the best attainable reduction by CO gas is

$$6C + 6O = 6CO \text{ (at the tuyeres)}$$
$$Fe^2O^3 + 6CO = 2Fe + 3CO^2 + 3CO$$
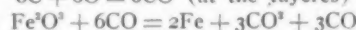
If some reduction is performed directly by solid carbon the reaction is

$$Fe^2O^3 + 2C = 2Fe + CO^2 + CO$$
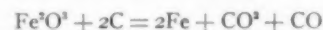
It is thus evident that carbon used directly in reduction performs three times as much reduction as carbon first burned at the tuyeres to CO gas. In the usual case, first given, it is even more efficient. The total carbon requirement of the furnace, therefore, decreases rapidly as soon as it becomes possible to use carbon for direct reduction, a condition which becomes possible as soon as the "equilibrium point" has been passed.]

III.

With any given furnace, the heat available in the smelting zone can be increased by increasing the temperature of the blast. If the charge is such that Grüner's ideal working applies to the furnace, then increased temperature of the blast does not disturb that condition until the "equilibrium point" is reached.

After the increased heating of the blast has brought the furnace to the "equilibrium point," further heating of the blast will allow of departure from Grüner's ideal, and with increasing economy of fuel the further the departure.

With any given furnace, the heat available in the smelting zone is increased by removing moisture from the blast. If the charge is such that the furnace is normally at the "equilibrium point," any increase of available heat allows departure from Grüner's ideal with resulting economy of fuel.

Removing moisture from the blast of a furnace at the "equilibrium point" will allow of increased use of solid carbon in reductions with increased economy of fuel.

It has been proposed to send hot blast into the furnace in which some oxides of nitrogen were placed. The splitting up of the nitrogen oxides would *increase* the available heat.

If a furnace were at the "equilibrium point," introducing nitrogen oxides with the blast would allow of departure from Grüner's ideal, with resulting economy of fuel.

Finally, auxiliary heat may be introduced into the smelting

zone of the blast furnace electrically. If the furnace is work-
ing under the *regime* of Grüner's ideal,, such electrically intro-
duced heat will *tend* to bring it toward the "equilibrium point."
Until that point is reached, however, Grüner's ideal holds true
for the furnace.

When the equilibrium point is thus reached, further introduc-
tion of electrical heat enables a rapid departure from Grüner's
ideal; the more electrical heat used the less carbon need be
burned to CO gas, and the more carbon can be consumed by
direct reduction, with resulting economy of fuel.

At the limit, we can easily suppose *all* the necessary heat
for smelting purposes supplied electrically, in which case *all*
the carbon charged would be used in direct reduction, and
this with great total economy of fuel.

*Application of electrical heat to a furnace at the "equilibrium
point" will enable the furnace to rapidly depart from Grüner's
ideal working with great resulting economy of fuel.*

Résumé.

It follows from the above discussion that Grüner's ideal
working is true only under certain conditions, which are, how-
ever, usually present. Under changed conditions the furnace
reaches an "equilibrium point" beyond which Grüner's ideal
working is inconsistent with economy of fuel, and greater
economy of fuel is attained the further Grüner's ideal can be
departed from. Heating the blast, drying the blast, faster
working, introducing nitrogen oxides or using electrical heat
in the smelting zone are means for reaching the "equilibrium
point" and eventually of working in the region past that point
where *Grüner's ideal no longer applies, but is a false principle*
leading away from maximum economy. In this region, greater
economy of fuel is attained the further Grüner's ideal is de-
parted from.

Lehigh University.

A New Process (Spray Process) for the Production of Metallic Coatings.

By M. U. Schoop.

There are known quite a number of different processes for
the production of metallic coatings, like the electrolytic or
electroplating methods, the "hot galvanizing" method and others.
All these require that there be a metallic foundation surface on
which the desired coating is produced. Other inherent disad-
vantages, especially of the electroplating methods, are that the
process cannot be carried out at any desired place and that the
size of the articles to be plated is restricted, in view of the given
location and the given dimensions of the plating tanks.

The following notes will show that these disadvantages are
directly overcome by my new spray process for the production
of metallic coatings. I have described it in a paper read on
April 13, 1910, before the Engineers' and Architects' Society
in Zurich and the process was also discussed by Professor
d'Arsonval on April 19 before the Academy of Sciences in
Paris.

The essential features of my new process are that metal in
liquid and highly heated condition and in finely subdivided form
is sprayed onto the surface to be coated.

The fine subdivision of the metal may be accomplished in
very different ways. The method of subdividing or atomizing
the metal is itself not an essential point of my invention, or,
rather, discovery. For what I have discovered is a new tech-
nical principle, namely, that it is possible to produce from
"atomized" metal dense and workable metallic films which may
be made at will firmly adhering or detachable.

For the subdivision of the metal, gases or vapors can be
used as agents both for exerting a pressure on the metal and
for atomizing it. But the subdivision of the metal can also
be obtained by simple mechanical devices. For instance, a
method which does not use gases or vapors is to pass two jets
of liquid metal in continuous stream out of two nozzles in such

a way that the two jets meet each other under a certain suitable
angle. When they make contact they mutually tear each other
up. Further centrifugal nozzles may be used in which the jet
of liquid metal emitted under high pressure is atomized by
centrifugal force.

According to the experience so far available the easiest and
the most rational method of subdivision makes use of com-
pressed gases or vapors. A suitable method of this kind is
shown in the accompanying illustration. The metal, a coating
of which is to be produced, is molten in the crucible *e* and is
then forced out by means of a compressed gas, or air, or vapor,
so as to leave the nozzle *g* through a capillary opening.

Immediately after leaving the nozzle, it is acted upon by a
stream of gas or vapor of suitable cross-section and is thereby
subdivided so that a metallic mist is produced through which
the articles to be coated are passed quickly, the passage lasting
for one to three seconds.

One and the same gas may be used to exert the pressure by
which the metal is forced out of the nozzle and also to atomize
the metal. This is the case in the illustration where the gas
employed is contained in the cylinders *a* and is preheated in *c*
by means of the gas burners *d; b* and *b* are manometers.

In some cases, however, it is preferable to employ different
gases and also different pressures. For instance, to press the
metal out of the nozzle, a chemically inert or reducing gas may
be employed like nitrogen, etc., while for atomizing an oxidizing
gas may be used. By this arrangement it is, for instance, possi-
ble to produce lead peroxide coatings in form of firmly adher-
ing and strongly resistant films.

The different metals behave, of course, differently when
being atomized. The melting point is of importance, also the
greater or smaller degree of fluidity; further, the affinity of the
metal to the gas or vapor used for exerting the pressure and
for atomizing. The special devices for atomizing, the pressures
employed, the preparation of the articles to be coated, and quite
a number of other conditions are also of importance.

With certain metals care must be taken in selecting the proper
material for the crucible in which the metal is to be molten.
For instance, for melting aluminium and its alloys, iron or
steel crucibles cannot be used, nor crucibles of fire brick, porce-
lain or silica, since aluminium has a tendency to enter into
combination with materials containing silicon, and to form an
alloy.

Numerous tests have proven beyond doubt that when an arti-
ficial pressure is used, metals alloy much more quickly and
more easily than is the case under similar conditions at ordinary
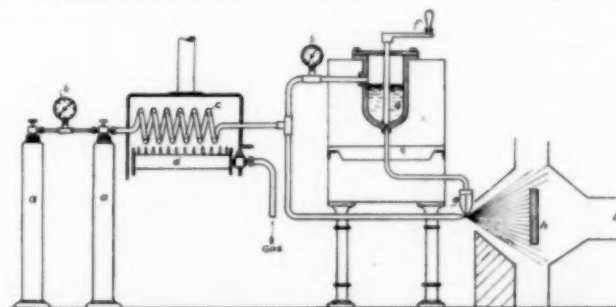
SCHOOP PROCESS FOR PRODUCTION OF METALLIC COATINGS.

atmospheric pressure. Thus I have found that ordinary gray
iron castings and steel castings easily alloy with aluminium as
well as with lead or tin if a sufficiently high pressure and a
correspondingly high temperature are employed.

The new spray process also permits to produce alloy coatings,
but the results so far obtained appear to show that a homoge-
neous subdivision in the spray and a homogeneous alloy coating
result only if the two metals are mixed in eutectic proportions.
If this is not the case, one of the two metals being in excess,
the lack of uniformity of the coating produced can be easily
established by means of microscopic examination. In other

words, my process gives a simple and sensitive method of testing the constitution and nature of alloys.

It is probable that coatings of alloys can be produced by my process if the same surface is simultaneously sprayed by two jets of different metals. It is also possible to produce metallic coatings in which other substances, especially hard ones, are mechanically included, thus producing "near alloys," such as have been recently discussed by K. Friedrich (this journal, April, 1910, Vol. VIII, No. 4, page 190).

A sensitive method of determining the quality and purity of the metallic coating consists in an investigation of coatings produced on mirror glass. For instance, coatings of pure tin produced with the aid of pure inert gases yield mirrors which can hardly be distinguished from the ordinary mercury or silver mirrors. On the other hand, when using compressed air or other oxidizing gases or in general a chemically active gas or vapor, the mirrors produced are more or less dull.

Microscopic investigation of metallic coatings produced by the spray process also permit to determine the size of the little metallic particles sprayed onto the glass—that is, to determine the degree of subdivision of metal in the spray method.

We may form about the following idea of what happens in the spray process. As soon as the stream of metal forced out under pressure leaves the nozzle, it is met by the vapor or gas stream under pressure. It is thereby torn up and changed into a form of mist. Each individual metallic particle when sprayed onto the surface to be coated loses its surface tension and its initial globular form and is pressed out into the form of a little circular film. All the little circular films produced in this way on the surface are automatically welded together into a continuous metallic coating.

When a pressure of 25 atmospheres is used, calculation shows that the metallic particles in the spray have a speed which is more than 20 km per second—that is, a speed which is 25 times that of a bullet from a gun used in the German army. This enormous speed explains why metallic particles while no longer fluid, when sprayed onto the surface, form a closely adhering and homogeneous film of surprisingly high density.

Under suitable conditions and with high gas pressures the density of the coating produced is not very different from the density of the metal under normal conditions. As an example, I mention the result of specific weight determinations which were carried out in the chemical and metallurgical laboratory of Bueler in Zurich. The density of tin, rolled or hammered, was found 7.47; that of cast tin, 7.29, and that of tin coatings produced by the spray method, 7.42. Other similar results were obtained for other metals.

The following fact is of interest: The zone of metallic mist is at a comparatively low temperature, about 40° C. to 60° C., although the molten metal may have a temperature of several hundred degrees and although the gas used for pressure and atomizing (nitrogen or superheated steam) is also at a high temperature. This fact, which was surprising at first, must be due to the sudden reduction of tension of the gas whereby the extraordinary cooling effect of the expanding gas also chills the finely subdivided metal with its enormous surface. It is, therefore, possible without any difficulty to apply the spray method for coating substances with a low melting or ignition temperature, such as paper, wood, celluloid, etc.

The hardness of the metallic coatings produced by my method when measured by the well-known Brinell test is essentially greater than the hardness of metallic films produced by other methods (casting or rolling). This is surprising. In tests carried out by the Bureau of Testing Materials of the Institute of Technology of Zurich, the mean hardness of cast films was found to be 9.6; that of films produced by my spray method, 14.2. This great hardness should be an advantage in the manufacture of cuts and electrotypes in which hardness and resistivity are mainly important.

Finally, it may be mentioned that microphotographic investigation of the structure of coatings produced by my spray method showed, in agreement with the results of specific weight determinations, that porous places or slag enclosures are absent. Where the object is simply to produce an ornamental effect or to make a surface electrically conducting a certain porosity of the coating would not be of great account. But in most cases the object is to protect the surfaces against chemical or physical influence, or against the effect of weather, and in such cases it is, of course, necessary that the coating not only be adhering, but also as dense and homogeneous as possible. This is the case for the metallic coatings of my spray method and they can be subjected to any mechanical treatment like pressing, polishing, etc.

The metallic surface to be coated is first to be freed from adhering dirt or oxides by some suitable process. Further, in many cases, especially where metals are to be coated, it has been found advantageous to preheat this surface before the other metal is sprayed on it.

Iron rods which had received a tin coating by the spray method were subjected to severe tests. In tensile strength tests, up to the breaking point, it was found that the tin coating extended to the point where the break occurred. Tortion tests up to the breaking point showed that the tin coating would not crack off. Great changes of temperature, like heating up to 50° C. and cooling at 25° C., did not exert any appreciable effect on the coating.

I may add that in order to produce a non-corrosive coating of tin, zinc, or lead on iron, it is not absolutely necessary to preheat the piece of iron if only the surface is pure.

There can be no doubt that by means of compact transportable spray apparatus (somewhat of the kind of vacuum cleaners) the possibility is given of providing any piece of iron of any size anywhere with a metallic coating for protection against corrosion—for instance, steel structures like bridges, cranes, etc. This would be a cheap and easy substitute for the usual paint coating.

Applications.

The fields of application of this method are so exceedingly manifold and relate to so many different industries that it is not possible to enumerate them here in detail. In the main there are two large groups of applications, namely, first, the production of adhering thin metallic coatings for ornamental purposes or for the protection of given articles, and, secondly, the production of detachable metallic deposits of a certain form. The deposit of metal on an insulating body in order to get electric conductivity, for instance, in the manufacture of electric condensers, belongs to neither of these two main groups. The same is the case with the production of parabolic and other mirrors by the spray method.

We will mention a few applications belonging to the first group, namely, the production of adhering thin metallic coatings to ornament or protect a certain article. One application is the metallic coating of wooden telephone and telegraph poles at that point where they are most liable to go quickly to pieces—that is, where the pole is inserted in the ground.

Another interesting new application is the covering of wood propellers with metal for aeronautic purposes, whereby the wood is protected effectively against the influence of the weather and besides the resistance due to air friction is very materially decreased.

In the case of balloons a very thin film of aluminium about 1/30th mm thick, applied by the spray method, is sufficient to make the material of the balloon air-tight and gas-tight.

In the case where bottles are to be closed so as to render refilling impossible and at the same time protect the contents from the air, a thin layer of tin applied over the top of the bottle by the spray method is cheaper and easier than the usual methods of hermetically closing the bottles. Paper metallized by the spray method may be used instead of tinfoil or aluminium foil for the packing of chocolate, etc.

The double problem of plating articles with aluminium and

of plating aluminium articles with other metals in an effective and durable manner is solved in a very simple and effective way by the spray method, and there seems to be no doubt that this method will open new fields to the use of aluminium.

As other applications of the spray method, I may mention the metallic coating of products of the textile industry, the manufacture of works of art, the lining of tanks and other vessels for chemical engineering purposes, the metallizing of wall paper, the metallizing of stage scenery, etc.

The second large group comprises the manufacture of detachable metallic deposits in certain desired forms. This class comprises the manufacture of cuts, gives a substitute for electrotypes and also includes the manufacture of hollow bodies, of seamless tubes, etc. It may be that the present electrotyping industry wil be seriously threatened by the competition of the new spray method, because the latter permits to make metallic copies of the original cut in fractions of a minute. The metallic copies thus obtained are in no way different from ordinary electrotypes, except that they are much cheaper. The manufacture of hollow articles of metal is also rendered simpler and cheaper and it may be expected that all industries which at present produce metallic coatings or metallic reproductions will feel in a near future the competition of the spray method.— *Zurich, Switzerland.*

The Elements of Slime Concentration.*

By WALTER McDERMOTT.

Introduction.

In considering the progress of concentration methods for the last 30 years, it appears possible to draw certain conclusions which may be considered as elements governing the probable developments in the future. To those who have been in touch with this branch of ore treatment it is very evident that there has been nothing revolutionary in the practice for the period mentioned.

There have been hundreds of inventions; there have been some dozens of commercial introductions of new machines, which were expected to be revolutionary, but which have had only a short period of favor, followed by disappearance from use; and there have been a very few new types which have survived the struggle for existence. The most striking fact has been the very wide and increasing use of two types of machines, the shaking belt and the jerking table.

Inventors, of course, will invent, and they will hope; but very often they will not investigate what has already been done, and those persons who have been in the position to hear of and examine many new inventions must be continually struck with the fact of recurrence of exploded ideas, and the existence of certain limitations which must be considered in a successful slime saver.

It is easy enough to save the coarser particles of any mineral with a specific gravity admitting it into the concentration class, so that invention is more prolific in the means of saving the finest particles. Experience all over the world, on all kinds of ore, has certainly demonstrated a number of facts which enable one to decide pretty accurately the chances of success for any new invention by a mere examination of the principle on which it works, and of its details of construction, without any actual trial on a given ore. Many think that every new invention must be tried before it is judged.

Even on actual new inventions this is only partially true, because life is too short and capital too limited for trial always to precede judgment; but in the vast majority of inventions the novelty is only a novelty to the inventor; the machine itself is an immaterial modification of something already tried and condemned, or it is opposed in principle to established conditions of success.

*A paper read before the Institution of Mining and Metallurgy in London on April 21.

The following notes are an attempt to put down certain elements which to the writer seem established by practice; and to draw certain conclusions which may be of service as indicating the most promising direction for improvements in the arrangement of old machines, or in the designing of new ones.

Definitions.

Although the chief subject matter of these notes is the recovery of slime mineral, some of the processes to be discussed cover a wider range of sizes than what are usually understood by the term "slime," and a number of machines are in present use which deal at one operation with this wider range. To restrict the field of discussion it is proposed to confine this paper to the treatment of material which has passed a screen equivalent to 40-mesh I. M. M. standard, that is, apertures of 0.0125 in.

As the functions and efficiencies of a number of well-known machines are to be referred to, it is desirable to point out that to-day there are practically no master patents in existence on the best known types of machines; that these types are all (with varying details) manufactured by a number of firms; and, therefore, it is possible to refer broadly to *types* of machines without being misunderstood as referring to *makes* of machines. The following classification is herein adopted as covering all the machines necessary to refer to for the purposes of this paper:

(1) Fixed inclined tables (including slowly traveling tables).
(2) Jerking tables (including bumping tables).
(3) Shaking traveling belts.
(4) Jerking or bumping traveling belts.
(5) Shaking side-inclined belts or tables.

CLASS I.—*"Fixed Tables"* are illustrated by various forms of frames, with wood, canvas, blanket or other surface; by the old Cornish buddle, and by various forms of revolving buddles and side-inclined non-shaking belts. The slow revolution of a buddle or belt, and the intermittent automatic discharge of a frame, do not affect the actual operation of concentration on the surface, so all are included in the class of fixed tables for the sake of simplicity and brevity.

In this class, all the machines depend for results simply on the greater resistance of the specifically heavy minerals to the wash of water down an inclined surface, as compared with the resistance of the lighter waste. The more perfect of these machines do not make a simple division between "concentrate" and "tailing," but they have to make an intermediate or "middle" product in order to yield a clean concentrate.

CLASS II.—"Jerking Tables."—This class depends for results on a combination of greater movement across a table and greater resistance to wash of water down the table of the specifically heavy particles, as compared with the motion and resistance of the lighter waste particles. For high efficiency this class, like Class I, must produce a middle product, because, in most cases, a hard line of division between concentrate and tailing is not possible.

CLASS III.—*"Shaking Belts."*—The operation of this class depends on a greater variety of conditions than need be considered in the preceding two classes; and a misunderstanding of these conditions has led frequently to hasty generalizations as to the requirements for most useful efficiency.

The shaking or vanning motion not only settles the heavy mineral on to the traveling surface of the belt, but it prevents the larger particles from clinging to the surface. The upward travel of the belt is in opposition to the flow of water; it tends to keep a bed of the coarser particles on the belt, and this bed checks the too rapid flow of slime material down the incline. The shaking motion, by keeping the bed from packing on the belt, allows of much less inclination on this than on a fixed table, therefore of less rapid current of water. This class makes no middle product, which fact is a limitation on some ores, but a great advantage on others, where only one class of concentrate is required.

CLASS IV.—*"Jerking Belts."*—The actual concentrating process on this is the same as in Class II, but it is carried out on a side-inclined traveling belt, going in the direction of the "throw" of the concentrate, and so the discharge is assisted and does not depend entirely on the "throw." This class makes a middle product also. In some modifications, owing to the nature and frequency of the jerking or bumping motion, a complication is introduced in the relative conditions as the settling motion approaches one of the conditions in Class III machines and has less of the jerking action so important in Class II machines. Such modification merges then into Class V.

CLASS V.—*"Shaking Side-Inclined Belts or Tables."*—This type of machine is not much employed at present, though it has been reinvented a number of times in different forms. A slowly revolving buddle with a shaking or bumping motion works in effect the same way as a side-inclined belt. The shaking motion yields some of the advantages of class III by assisting the settling of the slime mineral, diminishing the inclination of bed and lessening the water required on fixed surface machines; but a middle product is produced, and the longer settling surface and thickening effect of the upward travel of the belt in class III is not obtained.

Conditions Affecting Efficiency.

To understand the conditions necessary for a close saving of fine mineral, it is advisable to begin by examining the action of the simplest (and at the same time the most perfect) appliance in use; and this is the vanning shovel, or some equivalent tool, in the hands of an experienced man. The quantity of material operated on is small; time is of no object, and the action of the tool can be varied to meet the requirements of the moment, so that all the conditions are in favor of the most perfect work.

For proper work the shovel surface must be smooth; and it must be of a very slightly curved section in all directions. There are two distinct systems used with the shovel; the one for throwing up mineral which is not exceedingly fine, and the other system for separating the finest sizes of mineral from the waste. The first employs a motion which is in effect the action of the jerking machines of classes II and IV. The second movement (used for the slime mineral) is the washing back of the waste by a succession of waves of water from the fine mineral—previously settled on to the shovel surface by a shaking motion. This second separation is in effect, but in varying degrees, the action of classes I, III and V machines.

Passing by the question of the saving of the coarser particles of mineral, as being unimportant for the purpose of this paper, the second motion of the shovel may be further explained. To take it in its most refined work, it can be assumed that a sample of tailing, from a concentrator working only on slime, is under examination in order to see how much fine material is escaping. The shovel having its charge of sample (as flowing from the concentrator) is nearly full of water, with fine waste and fine mineral in suspension. No separation of waste and mineral can be at once made, but the first operation is to settle all the solid matter possible on to the surface of the shovel. This settling is assisted by a circular, steady shake of the shovel, which shake at the same time works down the heavier material, however fine, into actual contact with the surface and below the lighter waste. A rapid shake assists in concentrating the settled mineral at one spot.

When the elements of settling time, and settling motion, have sufficiently operated, a series of slower and wider sweeps of the shovel introduces the separating effect of gentle waves of the water, which gradually move the lighter waste, along the shovel, and leave exposed the "head" of the heavy mineral. In this final separating movement there is a combination of the shaking with the washing action, as facilitating the movement of the lighter waste over the surface of the shovel.

The perfect work of the vanning shovel is shown by the fact that it will always give from the most perfect commercial concentrator a definite head of the fine mineral which is going to waste. This perfection of work of the shovel being established, its conditions of working may be taken safely as a guide to the conditions necessary for close slime concentration in practice; and conversely it can be accepted that machines which ignore the conditions essential to good work by hand vanning give only very imperfect results in the saving of slime mineral.

The conditions which are essential to good work, irrespective of the skill of the operator, can be stated as follows:

(a) Sufficient time for settling.

(b) A smooth surface for final separation.

(c) A gentle movement of the washing water.

(d) A motion to assist in differential settling and in separation.

These elements deserve detailed examination as to their application to the construction and operation of concentrators of various kinds.

(a) TIME.—It is quite evident, if a very small sample on a shovel must be given a relatively long time to settle before any separation can be effected, that no machine will do close work on slime mineral unless the fine particles are given adequate time to settle from the water in which they are suspended.

As all commercial machines are continuous in operation this function of settling time becomes in effect the ratio of quantity fed to surface and rate of flow. It follows conclusively that if a new machine is produced for which a large capacity is claimed, while the settling surface is relatively small, no arguments based on motion or principles of construction will meet the immediate criticism that an essential element of success has been ignored by the inventor.

It is true that there is a point of difference between a continuous flow of pulp over the surface of a commercial machine, and the settlement of solid particles from a nearly stationary charge on a shovel. This difference lies in the fact that a thin layer of pulp slowly flowing down an inclined machine surface will, by mere frictional contact of some of the particles with the surface deposit such particles. This fact, however, does not affect the general truth of the proposition that feed must be adjusted to settling surface and speed of flow, so as to secure the element of settling time.

(b) SMOOTH SURFACE.—What is well known to be an essential for satisfactory hand vanning—that is a smooth surface—has been proved by general experience in the use of concentrators for slime mineral to be also best in commercial work.

It is true that canvas tables and blankets are used under certain circumstances; but such surfaces cannot be considered in the list of perfected concentrators; they are either used for a special purpose (as in the case of blankets for the saving of fine gold) or as supplementary catchers of a little valuable mineral from the tailings of more perfect machines. It is a constantly recurring fallacy that a rough surface must be good for saving slime mineral. The fact that it is also good—and in fact relatively better—for saving fine waste quite annuls the supposed benefit of its retaining power on the mineral. If a clean concentrate is required the current of washing water must be so great, to prevent the waste clinging, as to involve loss of fine mineral; and if clean concentrates are not required, the machine is not a concentrator: it is only a contrivance, like a settling pit.

The history of past practice and an inspection of the most successful working of the present day show that the advantage of a smooth surface is generally recognized. Tables, buddles and belts have been made of wood, glass, slate, marble, cement, rubber, etc., all with a view to getting the necessary degree of smoothness.

The extraordinary clinging power of very finely divided mineral, when once settled on to a smooth surface, is seen in the action of any of the machines of classes I, III and V when an excess of wash water is turned on; for the lighter waste and the larger particles of mineral are driven off long before the finest mineral can be moved from the surface it has adhered

to. To free a smooth vanning shovel from fine mineral after a test, it has to be immersed again and again in water, or put under a strong stream. The possibilities of rough surfaces, varying all the way from coarse riffles in a launder to a smooth close canvas, have been exhausted long ago; and therefore a new concentrator, for which some form of rough surface is claimed as the chief advantage, can be safely left for time to dispose of.

The action of stratification within definite cavities by a shaking motion is quite different from the mere arresting effect of a rough surface above referred to.

(c) SPEED OF WATER.—The bad effect of an excessive rate of flow of water over a concentrator surface is twofold. A rapid current is, of course, in the first place unfavorable to the settling of fine mineral; and, in the second place, it is apt to refloat and carry off some of the particles which by rolling contact with the surface of the concentrator have succeeded in settling.

It is elementary that a too great inclination of a concentrator and an excess of water are both unfavorable to close work. It is unnecessary, therefore, to illustrate the effects of these conditions, combined under the head of "speed of water"; but the importance of the point is sometimes overlooked in the comparison of certain types of machines for certain work, and in the designing of new machines for obtaining large capacity.

(d) SETTLING AND SEPARATING MOTION.—It is difficult to explain to those unused to hand vanning how important is the shaking motion imparted, and in how many varying ways it can be used, guided by the eye according to how the mineral and waste are settling and distributing over the surface of the shovel. The whole success of the operation, and the clear exhibition of the "head" of fine mineral separate from the body of the lighter waste, depend on the way the settling is accomplished, and on the way gentle washing waves of the water are made to travel. A proper motion keeps the lighter waste in semi-suspension in the water, free to move over the clinging, sluggish mineral particles; whereas if both waste and mineral are allowed once to settle fully on the shovel (to pack) the waste cannot be moved except by such a strong wave of motion as is certain to pick up and move much of the fine mineral also.

In considering the effect of the shaking motion in actual concentrators in practice, it will suffice to take a machine of the shaking belt class, and note the effect of stopping the shake. The machine becomes at once of the class I type; it is a self-discharging frame. Its inclination must be increased, or water supply made much larger; its loss of fine mineral will be much greater, and its concentrates will be less clean than when running in its normal way on the same feed of material.

Another very important effect of the shaking motion is that unclassified material can be treated in a way not otherwise possible, both on the shovel and on shaking belt machines; and this will be referred to more fully below in treating of classification.

Direction of Probable Improvements.

If the above evidence and conclusions are accepted, it is perfectly justifiable to judge any proposed new concentrator by its agreement with, or departure from, certain principles. The design will doubtless fall within one or the other of the five classes described above; unless it falls into one or other group of certain well-traveled and exhausted lines, such as rate of settling in water, resistance to upward flow of water, centrifugal separation, separation by air, separation in a denser medium than water, and flotation processes.

Putting aside the last-named group (flotation processes) as not really in the nature of concentration by specific gravity, and so not within the limits of this paper, it is worth while pointing out one of the great obstacles to the perfection of a machine for saving fine mineral by any of the other principles just mentioned.

No system of practical classification has yet been invented which does not in the end leave a great distance in the sizes of particles contained in the final class, the class in which the great loss occurs in concentration. As a consequence, separation by rate of settling, by upward flow of water, by a denser medium than water, all become imperfect and are usually quite impracticable.

It is well known that air separation—already tried in dozens of different ways—is most unsatisfactory on dust, and it is usually inapplicable for commercial reasons.

Many have been the attempts to employ centrifugal force, which on first thought seems eminently adapted to the saving of fine particles by its power of intensifying the function of specific gravity, but in practice its application becomes impracticable. With such fine and unequally sized particles as are under discussion, a separation by mere form of trajectory is not possible; and, as a consequence, the action of a centrifugal concentrator must be almost necessarily one of packing the material against the sides of the revolving machine.

This action of packing, or building up on the walls, is in its very nature absolutely opposed to any continued separation of particles of differing specific gravities when once fairly started, and hundreds of failures witness to the apparently insuperable difficulties of making a continuous discharge to this class of machine. Attempts which have been made to combine a washing separation by water (requiring a freedom of motion between particles) with the packing action of centrifugal force seem to point to an inherent antagonism between the two forces which would require a delicacy of adjustment impracticable in practice.

From the above it seems probable that new concentrators which require serious consideration will be modifications of established types, and to meet the expectation of doing work comparable with that of existing machines they can pretty safely be judged by the extent to which they meet conditions which have been referred to.

Of course, all new machines are said to do wonderful work; but history shows how exceedingly difficult it is to get at comparable facts, and how very unreliable as a guide are the testimonials and figures of millmen in the matter of concentration. The great variety of practice all over the world, even on similar ores, in both the use and distribution of different types of machines, shows at once how difficult it is in this branch of milling to get rid of the personal element, and the limiting effect of special experience; and again, the influence of mere fashion is often apparent in the arrangement of mills.

It may be mentioned here that on ores from which a single concentrate is required (not, for instance, concentrates separate of galena and zinc blende from waste) it is not advisable to make a middle product. A middle product may be needed when the coarsest particles carry attached mineral and need regrinding; but on material finer than 40 mesh this is not often the case. Many concentrators do make middle products, and it is certain that losses are incurred, and are often obscured by the fact that the middle product must be redressed; for a secondary loss is thus involved, and must be added to the loss in the tailing of the first operation.

There are often good reasons in practice for using a middle-product-making machine, but it should not be overlooked that in principle, and for the conditions assumed, there is a definite objection to the system; and, therefore, in designing improvements over existing machines it is a point which will count in actual commercial results in the end. Questions of capacity, cost and expediency often influence the choice of the details of arrangement of mills; but the question under discussion is one of individual efficiency of work, which should be well weighed in the first instance. It is known that in modern concentration mills losses are often high; it is therefore no heresy to refer in detail to certain probable causes of loss in existing practice.

To sum up the conditions which seem from the above to be most favorable to the evolution of any new concentrator which may reasonably be expected to prove an advance over existing

machines, it can be said that there must be a large and smooth surface; a regular settling motion which does not jar the finest particles from their contact with the surface when settled, and a delivery of clean mineral without the production of a middle class. The provision of a proper settling motion insures the other necessary conditions of slight inclination of surface, and the minimum quantity of water in pulp and for washing purposes. For example: a revolving buddle, which as ordinarily used, will require an inclination on its surface of 1½ in. to the foot, can, with a proper shaking motion, work to better advantage with ½-in. fall, and will need less water.

Classification.

The necessity for classification before concentration is so marked in some cases and with some machines, that it is accepted by some people as a comprehensive dogma with full scientific warrant; but actual experience has shown very clearly that in certain cases it is distinctly undesirable, and in other cases it is so imperfectly applied as to largely neutralize its advantages.

Bearing in mind the limitation in this paper to the consideration of material finer than 40 mesh, it is a clearly demonstrated fact that machines of class III (shaking traveling belts) give better results both industrially and theoretically without classification. At intervals of some few years new teachers arise to point out that the treatment of an unclassified pulp is in contravention of elementary laws of scientific concentration on any machine. It will be worth while here to push this argument to a conclusion, to see what, if anything, science has determined, and what practice has demonstrated.

In what scientific formula has it ever been attempted to lay down the relative values of the functions of specific gravity, inertia, surface resistance, speed of settling, friction of grains, rate of water flow, and adhesion to machine surface, in the case of a mixture of particles from 40-mesh aperture to impalpable slime, on a machine of class III type?

Such a formula might amuse a mathematician; it would be beyond the comprehension of the writer and of most people, and it would carry no conviction whatever in a practical sense. It is only necessary to fall back on certain results and details of actual operation, to show that some of the functions above named are so different relatively on different types of machines, as to make conclusions drawn from one machine inapplicable to the results to be expected from another machine. An illustration will help to make this clear. If a silver or copper coin be put on a machine of either I, II or IV class, it will be delivered with the concentrate. If the coin be put on a machine of class III it will not be delivered with the concentrate. Any scientific formula which correctly accounted for these opposed results could never be construed into a dogma on classification in general, since the effect of the function of inertia is quite different in the two cases.

It is known that classification is not necessary for a sample treated on the vanning shovel; and as this treatment is taken above as a standard of work and as a guide to investigation, it would seem to indicate in advance that classification is not a fixed law of perfection in concentration.

Furthermore, the experience of thirty years, on all kinds of ores, in many different mining fields, has established most definitely the fact that there is a distinct advantage in treating unclassified pulp on machines of class III. The coarser particles form a bed on the belt owing to its upward travel; and the fine slime instead of flowing down in unchecked waves over the belt surface (as it would do if treated alone) flows more slowly, by a filtering process in part, and so has better chances for settlement of the fine mineral.

In practice, the blind adherence to the belief in classification has led to some curious results. A very unsatisfactory method of classification is usually adopted, that of simple pointed boxes. By this system the coarser or bottom discharge is of necessity accompanied by a stream of the finest slime; and this last goes to a class of machine not adopted to its treatment. In some modern mills, arranged especially with a view of keeping coarse and fine apart for separate treatment, the first step, that of classification, is thus most imperfect. A step in advance is made in some cases by the introduction of upward flow classifiers, with a disadvantage from the continued dilution of the stream of pulp for the final treatment. The experience obtained in some of these mills has led to the supposed discovery that machines of class III are not proper machines for slime treatment, and they are therefore given an intermediate product of the classification system.

This conclusion is quite opposed to general experience; because machines of class III were originally developed for, and are continuously used for, the direct treatment of ores crushed fine in batteries, and containing small quantities of finely divided and brittle minerals, which are valuable enough to demand very close saving. Such ores, crushed under such conditions, must need close saving of slimes, and must make much slime value, and it would be a remarkable thing if a class of machine which is so successfully used for such work should be now discovered to be not adapted to slime saving. The explanation of this erroneous conclusion is simple, namely, that by attempting classification, and then experimenting with the very diluted slime alone, the machines of class III are not working under the best conditions; and they have, furthermore, very little capacity when so improperly limited as to size of pulp and when so overcharged with water.

In most of the large modern mills of the United States and Mexico, machines of class III are used for the finest sizes, and the slime losses due to imperfect classification already referred to are sometimes diminished by a retreatment on machines of class III of the tailings of class II machines. In some of these mills the finest overflows of classifiers or thickening boxes are run into settling pits, and treated as a concentrate for addition to richer products for the smelter; but such treatment is not available when clean products are essential to subsequent treatment, for example, with tin ore. Of late there has been a tendency to the use of fine screens in place of some of the water classifiers in a mill, to this extent avoiding the slime discharge of the ordinary classifier with the coarser particles, while making a separation by size alone, instead of by equal falling particles.

It is a very usual thing in mills to treat the tailing of one slime machine by another machine of the same or different construction; for example, buddles are often made double, or a convex surface may be followed by a concave one; and frames or canvas tables may be used in a long succession, each saving a little.

This double treatment can be used with great effect on machines of class III, treating unclassified pulp, but is very seldom attempted. The loss in the tailing of the first machine is in part saved by the second, and in some cases (for instance, with tin ores) where the pulp from the stamps is fine enough to go direct and unclassified to the first machine, the double treatment will be most effective as compared with a classification and treatment on several types of middle-product machines.

To get the best results from the second machine, it is advisable to use a large pointed box between, to get rid of some excess of water introduced by the first chamber; and an increase in belt surface on the second row will add to the efficiency. By this system the slime mineral has a double chance of being saved; there is no excessive dilution of the slime; clean concentrates can be produced, the losses due to rehandling of middle products are avoided, and the whole plant is the most simple and automatic in character.

If the tailing of the second row of class III machines contains enough fine mineral to be worth additional treatment, a properly arranged classifier will deliver this mineral in the fine overflow and in a condition suitable for treatment on revolving buddles or frames, or other wide surface appliances suitable for poor material.

When a screen is used in the battery too coarse for effective work by machines of class III, an additional finer screen can be used to take out one or more sizes of particles for machines of class II, or even for jigs.

* * *

In the discussion which followed, several speakers emphasized that the paper treated not with the concentration of slimes alone, but with the concentration of slimes with fine sands. It was also pointed out that where purely mechanical concentration failed other methods, like electromagnetic or electrostatic separation, or flotation processes, might help. But the author's paper was restricted to a discussion of mechanical separation by means of water.

The advisability of mechanical separation was also discussed by various speakers. Two speakers also expressed the hope that a method of concentration, based on centrifugal force, may still be developed.

Mr. H. L. SULMAN said that, considering the four factors alleged by the author to govern the deposition of slime-material. viz.: (a) Sufficient time for settling; (b) a smooth surface for final separation; (c) a gentle movement of the washing water; (d) a motion to assist in differential settling and separation; he thought they would be in general agreement as to the value of the first three for slime deposition. In regard to (d), however, this seemed a more debatable point; in most of the recognized types of mineral slime-savers that factor was absent.

Mr. Sulman thought that the majority of recognized slime-concentrators were included in the author's class I. Concerning the class III type, he said that it was a machine which had the undoubted advantage of dealing with a wide range of particles, often working better without the previous close classification demanded for the best results with other types of vanners, its bed of coarser concentrates acting as an intercepter for some of the slime mineral. It was admitted that vanners of this type III saved more slime-mineral than did machines in other classes, which dealt equally with concentration of coarser particles from sands. But he doubted that on slimes alone (i. e., without a bed of coarser concentrates) the class III type would equal some in class I as a saver of fine mineral particles.

Mr. Sulman also remarked that the class I machines, which gave the closest savings of slime-mineral per se, were provided with no differential motion. He concluded, therefore, that the author's contention as to factor (d) was not always borne out in actual practice; or it might be that the ideal machine, in which the closest slime-concentration was to be in combination with a different motion, had yet to be invented.

Mr. W. R. THOMAS spoke on the concentration of tin ores in Cornwall. They now had, in Cornwall, examples of classification before concentration and concentration before classification; in the former, in some cases, the spigot going on to the jerking tables and the overflow to vanners, while in the latter case the whole was taken on to the vanners; of one instance he knew, where the whole mill product, after going over the vanners, was classified into sands and slimes, and the sands were put over a jerking table. In Cornwall, they used either simple pointed boxes or upward current classifiers. His experience during the past few years had been that where the latter form of classification was in use, by varying the amount of water they could vary the character of the pulp on both the tables and the vanner; if one increased the amount of water which was coming from the upward jet, one could throw sands onto the vanners.

He had noticed that the tin ore recovered by treatment of the middle head from the jerking table was much finer than that recovered from the head from that table. The vanning shovel also showed that a small quantity of similar sized hard-grained tin was found in the tailings from the first machine.

The catching of the coarser particles of mineral was not a difficult matter, but to catch the finer particles contained in the slimes was a most serious problem. Too close a concentration

was likely to increase losses. In some of the best cases, where there was comparatively little associated mineral, the practice was to concentrate it up to 25-35 per cent black tin, which on a pulp having an assay value of 28 to 35 lb. a ton meant a concentration of 20 to 1. To attempt to procure a clean concentrate would appear to result in a loss which otherwise might be obviated.

It was probable, however, that the main proportion of the loss was confined to the slimes, and it also appeared to be evident that both tables and vanners, with or without existing methods of classification, had failed to catch a fair proportion of the mineral values contained in the same. There, then, was the field for those studying the subject-matter of the paper, slime-concentration. The old school, who were responsible for, and who had so great an attachment for, the vanning shovel, originated the idea of the fixed hand frame, which had since given way to the automatic frame. They saw the necessity of running the slimes in a thin layer over a large area of inclined wood frames which secured an even flow, and where an acceleration or diminution of speed did not take place, as was the case with either the concave or the convex principle applying to both buddles and revolving tables. Owing to the comparatively low value of the slimes step concentration became a necessity. Many of the slimes were worth only perhaps 3 or 4 lb. of tin per ton. That had been followed up by applying a revolving table, either concave or convex, or a combination of both, for the treatment of the pulp from the frame, from which latter it was possible to obtain a fairly high grade concentrate.

The application of such settlers, referred to by the author for dealing with slimes, was very useful, and had to a large extent dispensed with the old slime pits, which were dirty, beastly things, and which took up so much space.

There appeared to be every reason to believe that a slime plant of that description, with automatic frames and revolving tables, systematically laid out, would catch a fair proportion of the tin escaping from the modern concentrators now in use, and that at an extremely low rate of working cost.

Mr. R. E. COMMANS thought that the types of machines best suited for the concentration of slimes and fine material are not those having a jerking or bumping motion, but machines of the shaking, traveling belt type—that is to say, machines on the lines of the well-known Frue vanner. When a small percentage of very fine material has to be separated, canvas tables of self-acting slime frames of large area, over which the pulp can be distributed in a thin film, might, as suggested by Mr. Thomas, be used with advantage, but in most cases it will be found that a sufficiently high percentage of the rich mineral can be secured if the de-watered pulp from the first set of vanners is passed over a second set.

Mr. S. J. SPEAK pointed out that the paper explained why different types of machines possessed particular advantages, and this might be amplified by two examples, from his own experience. Both cases were gold mills, working on sulphide ores; in one case good concentration rendered the tailings too poor for cyanidation; in the other case this was not so, but it was desirable to make a concentrate in order to save the galena in the ore. The first mill was fitted with eight frue vanners and two jerking tables; and experience proved that it was best not to classify before concentration, and that better results were obtained by discarding the jerking tables altogether. The second mill had only Frue vanners, but as it was desirable to throw as much zinc blende and pyrites into the tailings as possible, they were obviously not suitable machines for the work, and jerking tables would have been much better.

Dr. O. J. STEINHART emphasized that the broad line underlying the whole principle of concentration was that of equal falling particles in water, and stood up for Rittinger's law.

Mr. W. S. WELTON emphasized the necessity to ascertain what the actual losses in slimes concentration amount to under present conditions. For a number of years he carried out a

series of experiments with a view to discovering the causes and amount of loss in concentrating ores. He found that from 8 to 20 per cent of the tailings were carried off by the water employed, and that this portion was very much richer than the sands recovered as tailings. In consequence, the assays of tailings gave a very imperfect idea of the amount of loss in concentration. The collection of the float tailings on a large scale was found to be impracticable, and the actual loss had to be calculated as the difference between the assay value of the feed and the values recovered, divided over the weight of feed, less the weight of concentrates. It was then found that the loss, calculated as a percentage of the assay value of the feed, was as follows, in a concentration to four times the value of the feed:

With a silver ore sands, 38 per cent of the value of the feed.
With a silver ore slimes, 55 per cent of the value of the feed.
With a copper ore sands, 29.7 per cent of the value of the feed.
With a galena ore sands, 16.98 per cent of the value of the feed.

It was found that the percentage of loss on the feed remained the same for any given degree of concentration of the values, whatever might be the value of the feed; but increased with greater degrees of concentration, when there was no regrinding of the ore during concentration—but with regrinding of the ore there was a considerable addition to the loss. The results in concentration determined experimentally by him were confirmed by the results of concentration on a large scale, and again, recently, by independent experiment.

Mr. J. H. COLLINS referred to the treatment of the impalpable slime of the tin mines. The only successful method of separating the particles of tin-oxide from the waste in true slimes has been by standing them upon flat and slightly inclined surfaces, from thin films of flow, without shaking.

The plant commonly employed with this object, consisting mainly of the well-known "self-acting" and "revolving" frames, is cheap in operation and moderately effective, but somewhat costly in erection and inconvenient.

The author, Mr. WALTER McDERMOTT, in his reply, said that while he was referring to the difficulties of saving slimes, since they all admitted that the coarser particles could be saved, it was saved in certain cases in conjunction with fine sand. Part of the argument he intended to bring out was the question whether classification in certain cases was advisable, and whether machines of a certain type did not do better work when there was some finer sand mixed with it, and also that those same machines, by a double treatment, would diminish the loss which every machine must make in practice.

Mr. McDermott then replied to those speakers who had referred to frames and other simple appliances being used after other machines. He considered that that was no argument whatever, because if one used the other machines again after the frames one would still save something. Because one machine saved something from the tailing of another, that was no proof that the second machine was better than the first, because one could reverse the order with similar results, and put it again over the first machine and still save something. So that the argument drawn from the use of a simple machine like frames or blankets, or anything of that sort, after other machines, did not prove anything as to the relative adaptability of the two kinds of machines for saving fine slimes.

When large surfaces were required, treating poor material, it was natural that the cheapest kind of machines would be used. To run it over a series of frames over a large surface, and to spread the layers out thin, was, of course, to facilitate the saving of a certain amount of fine material; and, therefore, those simpler machines were always used after more complicated machines, although people did not think of using them to begin the concentration with; they used them simply as auxiliaries.

His argument was that it would be better to use a superior machine twice, rather than go to a less perfect machine which had the disadvantage of making an impure concentrate, and that, too, with the disadvantage often of a middle product. That particularly held in the case of material like tin, of which Mr. Thomas had spoken. It did not hold in the case of some copper ores, where the cost of smelting and the conditions as to fluxing enabled one to treat material as a concentrate, which, talking simply in abstract terms of close concentration, would not be considered a concentrate. Of course, if one ran the product of a stamp mill into large holes in the ground, one would save 95 per cent or 99 per cent, but it was not concentration.

Mr. McDermott also replied to Mr. Sulman, who had thought that machines of fixed surface (class I) are best for treatment of pure slime, because they are in general use for this purpose; and who had argued from this that a differential settling motion is not necessarily an advantage for a slime concentrator. One object of the paper was to dispute the correctness of some modern practice, so that the appeal simply to practice is not convincing; and furthermore, American practice does not support Mr. Sulman, because class III machines are very commonly used in preference to class I for slime.

In the paper it is suggested that while class 3 machines are preferable to those of class I, even on pure slime, the former machines will work to still better advantage if some fine sand accompanies the slime. There is a sufficient reason for the common use of fixed surface machines in the fact that a large surface must be used for efficiency in any slime treatment, and economy in first cost of plant largely influences practice. The subject under discussion, however, is simply perfection of work in concentration.

Mr. McDermott finally stated that he had no intention to object to classification in general, but rather to point out that in some cases of direct fine crushing it is unnecessary and disadvantageous; that in other cases of coarser crushing it is better to take out one or two sizes by screens without further waste classification; and that even with coarse crushing, classification may be carried further than is either necessary or advisable.

Standardization of Assay Values.

The committee for the standarization of assay and bullion values appointed by the Institution of Mining and Metallurgy has formulated rules for the guidance of assayers, metallurgists and mining men in reporting results of assays, metallurgical tests and similar operations where uniformity is desirable. This is for the purpose of avoiding ambiguity and of minimizing the possibility of misinterpretation or misuse of reports.

Reports of assay results, extraction tests, metallurgical and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the committee urges the necessity of definite statements as to the nature and condition of the sample as assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead and antimony, where both "dry" and "wet" methods of assay are in use, results should be reported as obtained by the "fire" assay or the "wet" assay, with further details as to method, if judged advisable. Similarly it should be stated whether the sample was assayed in the condition in which it was received, or air-dried, or dried at — deg. C.

In reporting that an assay for such a constituent as tin, arsenic or tungstic acid corresponds to so much "black tin," "white arsenic," "wolfram" or other recognized salable product, the report should give the assay result for such constituent, together with a definite statement as to the figures used in calculating such salable product. Thus, an assay showing 1 per cent of metallic tin might be reported as 1 per cent of metallic tin, equivalent, if all could be extracted as such, to 1.5 per cent of "black tin," containing 66.6 per cent of its weight of metallic

tin. Similarly, in reporting results where assay values might be mistaken as representing available produce, it should be stated that such are merely "gross" assay values, a statement of the actual "commercial" values being added whenever possible.

In such matters as ore-dressing, ore-treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and in its relation to the other samples, the committee emphasizes the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labelling, sealing and otherwise insuring the correctness and authenticity of samples submitted for tests and assays.

The following recommendations are submitted for adoption by the Institution:

(1) Assay reports shall state the exact condition of the sample when assayed.

(2) Assay values of gold and silver ores and products shall be represented in pennyweights and decimals, or in ounces and decimals, and not in ounces, pennyweights and grains. They shall be expressed in terms of fine gold and fine silver respectively, and not as "bullion."

(3) Assay values of alluvials shall be reported in grains and decimals of a grain of "fine" gold, or in pence (at 2d. per grain of fine gold), or cents per cubic yard. It is recommended that in the absence of specific information, 1 cu. yd. of ordinary alluvial, excluding boulders, be taken as equivalent to 3000 lb. (1.5 short tons).

(4) In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution. In the case of cyanide solutions, the use of the "fluid ton of 32 cu. ft." is recommended. It closely approximates to 2000 lb. and is in common use.

(5) When it is necessary to state or estimate the money value of an ore, etc. (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

(6) Laboratory sieving tests shall be made with the I. M. M. standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

The following recommendations on weights and measures have already been adopted by the Institution:

(a) The word "ton" shall represent a weight of 2000 lb. avoirdupois (29,166.6 oz. troy). It is advisable to abandon the use of the words hundredweights and quarters, and to express fractions of a ton in pounds or in decimals of a ton.

(b) The word "gallon" shall represent the Imperial gallon measure of 10 lb. of water.

(c) Temperatures shall be expressed in degrees Centigrade.

(d) Returns of gold and silver shall be expressed in terms of fine gold and fine silver respectively, not as "bullion."

(e) Gold contents of ores, etc., determined by assay, shall be expressed in money values as well as by weights; and in this connection the value shall be taken (as a convenient constant) at 85 shillings, of $20.67 U. S. currency, per troy ounce of fine gold.

All of the foregoing recommendations are meritorious and worthy of general adoption. For the most part they are already observed by the more careful assayers in the United States.

Hypochlorite for Water Purification.—In the *Engineering Record* of June 25 Mr. Francis D. West remarks that we appear to be rapidly approaching the "age of hypochlorite." Since the successful test of Fuller and Johnson, at Boonton, N. J., several cities have adopted this method of purifying their water supplies. It is excellent as an extra precaution, but is not a permanent substitute for a carefully operated filter plant. But the investigation of the subject is not yet exhausted. Mr. West urges further study of the use of hypochlorite with rapid uncoagulated filters and further suggests that the test of oxygen consumed is of value to operators of water disinfection plants.

Natural Gas and Its Production.

By Harold A. Danne.

To an Australian in the United States, the phenomenon of naturally generated gas issuing out of the ground ready for use is one of the most remarkable sights that can be seen.

This natural resource wherever it occurs is gradually converting many districts from second-rate farming communities into thriving cities abounding in wealth and manufactures and all that such features mean to the cosmopolitan population attracted by them.

West Virginia abounds in widely distributed bituminous coal fields and in the petroleum industries, and particularly illustrates the preceding features, inasmuch as in this State are found some of the steadiest gas wells and thermally richest gas.

Without going into the history of this natural gas and its discovery in the West Virginia fields, which involve George Washington and some of his contemporaries, it suffices to say that after the pioneers had drilled the first few shallow holes and had overcome the very great obstacles due to imperfect drilling rigs, reels and tools and ignorance of geological conditions, the industry has almost continuously expanded, particularly since the introduction of long pipe lines connecting the fields with refineries and with large cities, such as Cincinnati, Pittsburgh, Cleveland, and numberless small ones, and the extension and exhaustion of the products are now threatened in a period variously estimated at from 20 to 50 years.

Australians are accustomed to the use of artificial gas at about 4 shillings per 1000 cu. ft., and regard it at even that high figure as economical and useful, so that the wonder is in the West Virginia fields that at about 2 pence per 1000 the gas will not be exhausted the sooner, more particularly since it is serving the various purposes of hundreds of large factories and 15,000,000 to 20,000,000 people.

With the advent of the modern high-efficiency large-powered gas engines in increasing numbers, the amount of gas used for generating power is also becoming a very serious factor.

As an important feature in this use of the gas it should be borne in mind that at the well mouth the gas is saturated very often with light hydrocarbons, such as gasoline, having passed through or generated with, in very many cases, the strata known as oil-bearing pools. It, therefore, gives a very high thermal efficiency, some of it running as high as 1134 b.t.u. per cubic foot.

When it is compressed, either purposely or during the pumping operation carried on to force it along some of the pipe lines, it deposits its rich contents, and its thermal efficiency is decreased thereby until when it reaches some of the distant cities it gives often only 900 b.t.u. per cubic foot.

Some operators who have purposely compressed the gas have stated that it is neither reduced in volume nor in thermal efficiency, which might be explained by saying that their measurements and other data were faulty. The apparatus used in this gasoline extraction industry consists of double coils of piping similarly built to the old-fashioned ammonia condensing coils. Through the inner coil the natural gas compressed at about 200 lb. to the inch is allowed to pass. Thence it passes into the space between the inner and outer coils after having been duly cooled and the reduction of temperature due to its consequent expansion in many cases amounts to 50° Fahr. and so affects the molecular state of the gas that the hydrocarbons are deposited, one might almost say pressed out, in considerable quantities. I have it on authority of one operator that this juice, or fat, as it is called, often amounts to 2 gal. per 1000 cu. ft. of gas at 72° Fahr.

The natural pressure, or rock pressure, as it is often called, has been turned directly to account in running steam engines and pumps without combustion of any kind. This is the more feasible as such pressure varies from 500 lb. to 1200 lb. per square inch.

A peculiar phenomenon is noticeable when an engine so driven

is reversed. The lack of resiliency of the gas makes action so quick as to be almost uncanny, and if not carefully done is liable to strain the working portions of the machine.

A State law has recently been enacted prohibiting these wasteful methods and also compelling the shutting in of wells not in use.

A fact well known to all gas operators is that the natural pressure all over the States invariably falls about 25 lb. whenever the wind is blowing from the northwest. Why, no one has yet explained.

In the natural wells, i.e., in wells which were not artificially drilled, the gas pressure almost invariably rises very greatly at or before seismic disturbances and was particularly noticeable before the earthquakes of Martinique and San Francisco. This fact should be borne in mind when the theory of production of the gas, as formulated below, is considered.

The gas closely approximates marsh gas, CH_4, in composition and requires about 11 cu. ft. of air for the complete combustion of 1 cu. ft. of gas.

It has been estimated that the entire output of this field is close on 1,300,000,000 cu. ft. per day, of which about 500,000,000 is used on or about the fields, as much more is sent out and the balance is wasted in very many ways.

One of the least reprehensible ways of wasting the gas is in the manufacture of carbon or lamp black. This is only to be excused when it is considered that otherwise the gas would be totally valueless at present owing to its distance from the main pipe lines, or if the well be shut down, gradually fill in and the flow would cease.

This carbon black is manufactured by burning the gas beneath iron funnels and introducing cold-iron plates into the flames on which the soot or black is deposited. This goes on day and night and the plates are scraped by various methods, the result being collected and sold.

In the glass industry, which uses an immense amount of gas, the molten glass is heated entirely by natural gas, making glass of a peculiarly good quality. It is also clean and efficient and cheap to install.

Another industry is that of zinc smelting, in which the clay retorts are heated by the same fuel. This industry very often fouls the air for a considerable distance around with arsenical and antimonial fumes and also with uncondensed zinc, and the manufacturers assert that no way so far has been found to recover these waste fumes, although where the Federal Government has put pressure on firms down in the South the corporations have been forced to adopt more scientific methods in this respect and have found that thereby they have increased their profits as well as decreased the nuisance.

Natural gas is also very useful in the manufacture of bricks and pottery, particularly so as strata of suitable clays are often found in the vicinity of the wells. By varying the gas flames in the kilns the product can be graded in color from the lightest Mackinaw shade to deep chocolate brown, possibly by varying the surface composition and combination of the silica with aluminium and iron.

The manufacture of electrodes is also very largely entered into. In this industry crushed coke mixed with oil-stills residues is pressed out and carefully placed in small kilns and the whole brought to a red heat for various periods by means of natural-gas blasts. This roasts the electrodes and gives them high tensile strength when used in furnace work at great heat.

Before closing it might be appropriate to mention something of the method of generation of these natural hydrocarbons.

In Bulletin 410 of the U. S. Geological Survey, George F. Becker reviews the evidence. In it Mr. Becker finds the combination of hydrocarbons and igneous action so often that all suggestion of fortuitous collocation seems out of court.

One widely accepted opinion is that of Mendeleef's theory of the decomposition of the carbides. The strongest reasons against this is that natural carbides have never been found even in the regions where hydrocarbons are most abundant. Mag-

netic declinations, however, due to iron deposits, are often found. Whether or not this is a proof of the carbide theory as some investigators have claimed is open to question.

The writer has proposed a theory which men on the field have pronounced at least probable, and it may be useful to tabulate the facts before we consider the theory. They are as follows:

That oil and gas are accompanied by strong evidences of igneous heat.

That they are accompanied by strata of coal.

That they are accompanied by layers of salt water, which salt water was really one of the original reasons for their discovery.

That they are also accompanied as low as 2000 ft. down by beds of water-worn gravel and shells.

That some of the supposedly exhausted gas fields have recently been redrilled and have yielded paying quantities of hydrocarbons.

That natural gas on pressure and cooling deposits liquid hydrocarbons.

That as previously mentioned they are affected by seismic disturbances.

From facts such as these the writer proposes the theory that all these hydrocarbons are generated from natural carbonaceous deposits, such as bituminous and other coal, by the natural heat of the earth's interior brought into proximity to such deposits by seismic disturbances, which action is still going on under enormous pressure and the hydrocarbons are being generated under the, among others, liquid pressure of the streams of water overlaying them, and on the release of some of the pressure and changes in the temperature of the hydrocarbons so generated, some of their contents are deposited in the dips of the strata and in the pockets known as oil-bearing pools.

The dips, fissures and tumbled nature of these strata would account for the eccentric behavior and irregular location, comparatively speaking, of these gas and oil aggregations.

In Kentucky there are wells and caves known as "breathing caves," from the fact that the well or cave is alternately giving out and taking in gases. It has been recently proved that the gases thus given out are explosive mixtures of hydrocarbons and air. The gases taken in are, of course, air. The cause of the breathing may well be brought about by the variable pressure of the water which seals most of these subterranean gases, and which is itself in a state of agitation, and shows that there is still violent internal action taking place among the strata containing this water, which is apparently advancing and receding to cause the so-called "breathing."

The different natural hydrocarbon centers may be roughly grouped as Russian, Californian, Texan, Pennsylvanian, which latter might also be called West Virginian. In all of these different bases appear in the hydrocarbons produced.

These different bases have been investigated and differentiated by Park and Worthing, among others, and the hydrocarbons constituting them have been investigated as to nature and structural formula.

In these three main sources of natural manufacture of hydrocarbons in the United States, namely, Texas, Pennsylvania and California, to locate still more accurately the main centers of generation, a suitable method is to find the group of wells giving hydrocarbons at the highest pressure and thermal efficiency. This method is the more likely to give correct results, as it is a well-known fact that the farther away from the source natural gas is piped above ground the lower is its thermal efficiency, and I hold it to be reasonable that the same fact is true below. Furthermore, as a general rule, the lower the pressure at which the hydrocarbons issue the lower is their thermal efficiency. In this manner a center of generation in each of these large fields is found to exist, whence the hydrocarbons have issued and are still issuing to be deflected and carried through considerable distances in the underground connected porous strata associated with all these hydrocarbons and known to drillers and operators as the various "sands."

These main centers will be the first to supply and the last to exhaust. Thus it seems the crust of the earth in some parts forms a cover with a liquid seal of salt water to approximately three vast natural gas-producer centers wherein the deposits of former ages have been and are being eternally distilled by the steady, well-modulated heat of the earth's interior for man's discovery and use.

Clarksburg, W. Va., and Melbourne, Australia.

Measurement of the Thermal Conductivity of Fireclay.

Bulletin 36 of the Engineering Experiment Station of the University of Illinois (Urbana, Ill.) contains an interesting paper on "the thermal conductivity of fireclay at high temperatures," the joint authors being Dr. J. K. CLEMENT, of the Technologic Branch of the United States Geological Survey, and Mr. W. L. EGY, of the Engineering Experiment Station of the University of Illinois.

Although the investigation was undertaken to solve a specific problem—the thermal conductivity of fireclay at high temperatures—the method developed is of general interest in other lines of work where the question of thermal conductivity is of importance.

The thermal conductivity K of a substance at a certain point is given by the equation

$$K = -\frac{Q}{A}\frac{dr}{dT}$$

where Q is the heat (in calories per second) flowing in unit time across the area A at the point in question. The area A is assumed to be perpendicular to the direction of the flow of heat at this point. The differential quotient $-dT/dr$ is the temperature gradient at the point in question—that is, the differential quotient of the temperature taken in direction of the flow of heat (dr being an infinitely small length in direction of flow of heat, $-dT$ the corresponding temperature drop).

The most accurate method of measuring Q is to generate a known quantity of heat by means of an electric heating circuit, in such a manner that all the heat generated must flow through the substance to be tested. If a constant quantity of heat is generated until conditions have reached an equilibrium, then the quantity of heat conducted through the material per second must be equal to the quantity generated per second. This method may be used with a heating coil either in a hollow sphere or in a long cylinder. The latter form was chosen for these tests because of the experimental difficulties arising in the former.

In using this method, the assumption is made that there will be no escape of the heat longitudinally at the middle of the cylinder; that is, the exact amount of heat generated by a centimeter length of the coil, taken at the middle, must flow out through the corresponding circular section of just 1 cm thickness. To avoid errors from this cause, the length of the cylinder must be several times greater than its diameter.

The test pieces were made into cylinders about 40 cm in length and 12 cm in diameter, with a hole through the center about 3.5 cm in diameter for the reception of the heating coil. Four longitudinal holes, about 3 mm in diameter, were made, in which thermo-couples could be placed for the measurement of the temperature.

Applying the above equation to the case of a cylinder, the heat Q generated by 1 cm length of the coil in unit time is expressed by

$$Q = 0.2394\, EI/l \text{ calories,}$$

where E represents volts, I amperes and l the length of the coil in centimeters. The constant 0.2394 is the reciprocal of the mechanical equivalent of a calorie expressed in joules. The area A perpendicular to the flow of heat at a distance r from the axis is $2\pi r$ per unit length. Substituting for Q and A in above equation, we get

$$K = -\frac{0.2394\, EI}{2\pi l r}\frac{dr}{dT}$$

Assuming that K is constant between the temperatures T_1 and T_2 and integrating, we get

$$K = \frac{0.2394\, EI \log (r_2/r_1)}{2\pi l\,(T_1 - T_2)}$$

where T_1 and T_2 are the temperatures at points distant r_1 and r_2, respectively, from the axis. For any given values of r_1 and r_2, the only variables in this equation are EI, the electrical watts dissipated in the coil, and $T_1 - T_2$ the difference in temperature between r_1 and r_2. Thus the expression may be reduced to

$$K = CEI/(T_1 - T_2),$$

where

$$C = \frac{0.2394 \log (r_2/r_1)}{2\pi l}$$

Twelve test cylinders were investigated, which had been made by the Laclede-Christy Clay Products Company, of St. Louis,
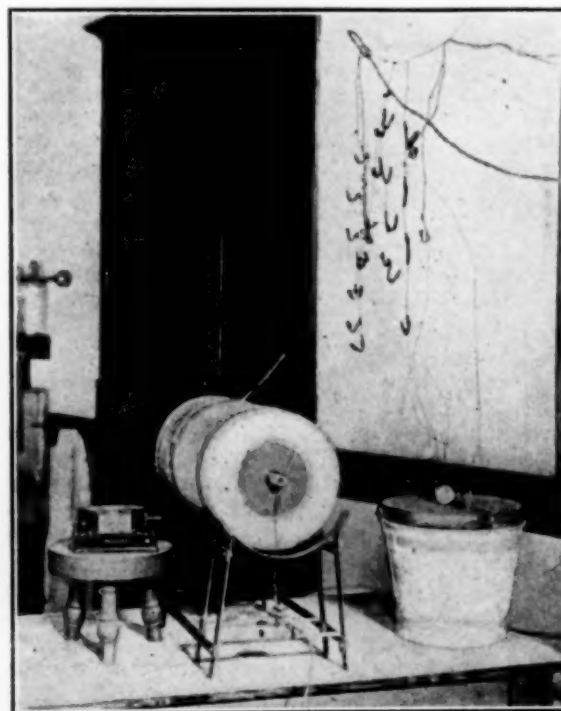

FIG. 1.—TEST PIECE SURROUNDED BY FIRE-CLAY JACKET.

Mo. They were made up from four different mixtures, which were marked A, B, 1 and 3.

Those marked A were of a dark reddish-brown color and contained no gravel. The structure appeared similar to sandstone. After heating in the test, these pieces were cracked in a great many places. The pieces marked B were also of a reddish-brown color, but were of medium coarse structure. They contained very small pieces of white gravel throughout the mass. Those marked 1 were a little coarser than the B cylinders and were brown in color. They contained a very small amount of gravel. Cylinders 3 were almost white and very coarse. They contained a large amount of gravel.

One cylinder of each composition was tested at temperatures ranging from 400° C. to 800° C., or 900° C. The remaining pieces were tested at only one temperature.

Fig. 1 is a general view of the apparatus with the test piece surrounded by the fireclay jacket, and with the thermo-couples in place.

Fig. 2 shows a longitudinal section of the furnace ready for use; aa is the test piece, surrounded by a large cylinder bb of fireclay, in order to get uniform radiation from aa, and also

to maintain higher temperatures. Coverings cc were placed over each end to prevent loss of heat in this direction. The whole was supported by an open framework of strap iron. A thermo-couple was placed in each of the holes D and F.

The heating coil was made of pure nickel wire about 1.8 mm in diameter, wound nine turns to the inch upon a ½-in. porcelain electric insulator tube. Commercial insulator tubes 20 in. long were taken for this purpose and the enlarged end cut off, leaving them 18 in. or 19 in. long. Small holes were chipped in the ends of each tube, and the ends of the wire pulled tightly through these holes, thus holding the wire in place. The holes were 16⅛ in. apart, allowing 145 turns of wire, or an equivalent length of 40.9 cm. Potential leads were fastened to the power wire where it emerged from the insulator, by winding one end of a piece of smaller wire tightly about it. The place where this connection was made had a rather high temperature, so nickel was used to prevent any thermo-electric effect at the junction.

The coil was at first supported in the cylinder by two small pieces of quartz tubing at each end (see Fig. 3). At the higher temperatures, however, the insulator softened and sagged down in the middle, and two additional pieces of quartz were placed under it, each one-third the distance from either end. The holes in the end of the furnace were packed with asbestos to prevent the escape of heat. The heating coil was made a few turns longer than the test piece to prevent cooling of the ends.

Current and voltage were read by Weston instruments. The current was taken from 110-volt mains and was regulated by means of two large rheostats. As it was necessary to maintain the current constant to one-tenth of an ampere for several hours at a time, and the line voltage varied considerably, a storage battery was placed across the mains.

The temperatures were measured by platinum-platinum rhodium thermo-couples, made by Heraeus. The e.m.f. of the couples was measured by a Wolff potentiometer and a galvanometer.

The heat which was generated at the middle of the coil flowed out along radial lines. This was tested by moving the outer couple F (Fig. 2) back and forth in a longitudinal direc-
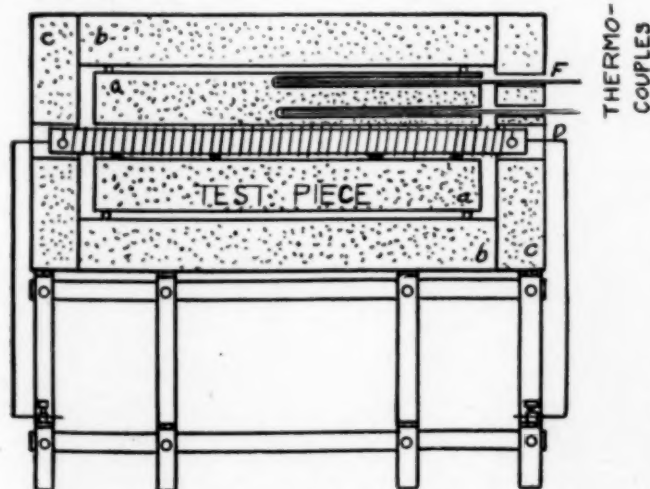
FIG. 2.—SECTION OF FURNACE.

tion and noting the reading in the various positions. This operation showed that the temperature was constant to 0.3° C. for a distance of 6 cm to 7 cm from the middle. As no heat will flow along constant temperature lines, there could be no longitudinal flow of heat in the 12 cm or 14 cm in the middle of the test piece, and hence the assumption is allowable.

The general method of operation was to start the furnace with a rather large current, about 25 amp, and gradually reduce this as the furnace approached the temperature desired. This took from three to five hours for the lower temperatures. The storage battery would then be placed in the circuit and the cur-

rent kept constant for two or three hours more. When the outer couple F showed a temperature constant to one-tenth of a degree, indicating equilibrium conditions, readings were taken. The voltage and current readings were taken before and after the temperature readings in order to be sure there was no change.

Each cylinder at the close of the tests on it was broken across the middle and r_1 and r_2 carefully measured in the plane in which the temperature readings had been taken.

The accuracy of the values for K in this work is limited to the accuracy in measuring r_1 and r_2. The temperature readings are certainly accurate to 1.0° C., and very probably to 0.5° C. The voltage and current readings are accurate to 1 per cent. An error may be introduced in the values of r_1 and r_2 because of the uncertainty of the position of the thermo-junction in the hole.
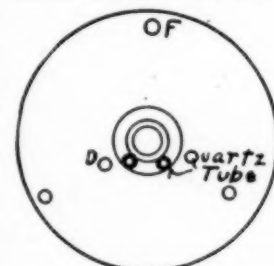

FIG. 3.—DETAILS OF QUARTZ TUBING.

If the couple were touching one side of the hole, it would take the temperature of that side, while r_1 and r_2 were always measured to the center of the holes. This error, however, would be a constant factor for the tests on any cylinder, since the couples were never disturbed after they were once placed in a cylinder until the completion of the tests on that piece. Thus the comparative values of K for any given specimen at different temperatures would not be affected by the values of r_1 and r_2.

The greatest error is caused by changes of current through the furnace. The supply voltage changed a great deal, and the batteries used were not large enough to maintain the current constant at all times. If the current should change just before a reading was taken, the temperatures would not correspond to the other readings, though they might not be changing at the time.

As to the results it is interesting to note that the conductivity of the two coarser fireclays was constant, while that of the other two, which were of finer structure, increased with the temperature.

For a test piece B the conductivity K (in gram calories per centimeter per second per degree C.) increases from 0.0021 at 350° C. to 0.0022 at 600° C.

For a test piece 3 the conductivity K was practically constant $= 0.00264$ between 400° C. and 700° C.

For a test piece A the conductivity K increased from 0.00245 at 500° C. to 0.0026 at 800° C.

For a test piece 1 the conductivity K was practically constant $= 0.0036$ between 400° C. and 750° C.

There were found, however, differences between the conductivities of cylinders of the same composition. These are probably due to differences in porosity. An investigation of the effect of porosity as well as composition of substances would, no doubt, lead to more definite results in regard to a comparison of fireclays.

The application of the above figures in practice is illustrated by the following example:

Knowing the thermal conductivity K of brick, the dimensions of the furnace, and the outside temperatures of the walls, the quantity of the heat transmitted through them may readily be calculated. Taking the specific case of a 210-hp Heine boiler at the University of Illinois, when working under full load—the area of walls exposed to the hot gases is about 364 sq. ft., or 338,000 sq. cm., and the thickness of the same about 20 in., or 51 cm. The average temperature of the inside of this area is approximately 1400° Fahr., or 760° C., and of the outside 150° Fahr., or 65.6° C., and $-dT/dr = 694.4/51$. If we now take the value of K as found for that test piece which was nearest like the brick in the setting—that is, $K = 0.0026$—and calculate the

heat conducted through the walls we have, using c.g.s. units:

$$Q = -KA\frac{dT}{dr} = 0.0026 \times 338,000 \times \frac{694.4}{51}$$

= 12,000 gram calories per second = 43,400,000 calories per hour.

In an appendix the method for calibrating thermo-couples for high-temperature measurements is described in great detail.

Continuous Pachuca Tank Agit tion at the Esperanza Mill.*

By M. H. Kuryla.

The agitation equipment of the new cyanide plant of the Esperanza Mining Company, El Oro, Mexico, consists of two batteries of six 14 ft. 10 in. x 44 ft. 8½ in. Pachuca tanks. The first battery of six tanks, locally referred to as the "Upper Pachucas," treats the flocculent slime produced from the coarse crushing in 120 stamps, and is worked on the "intermittent" or "cycle" system originally installed. The location permits of a gravity discharge to the storage tanks for the Merrill filter presses.

The six "Lower Pachucas" treat the granular slime resulting

Total possible agitation without filling or discharging loss .... 38.4 hours
Loss in filling and discharging...................... 5.1 hours

Net agitation .... 33.3 hours
Percentage of net agitation lost in filling and discharging, 15.3 per cent.

The yearly expenses connected with the operation of the three elevators figured out as follows:

	Mexican Currency.
Three elevator belts, each 12P′, at $6.............	$2,160
Power	1,825
Regular attendance	730
Buckets and bolts	35
Repairs, labor	1,095
Repairs, material (exclusive of belts)..........	500
Total per year	$6,345

Present Continuous System.

The difficulties and expense experienced with the elevators made it highly desirable to deliver the pulp by gravity to the
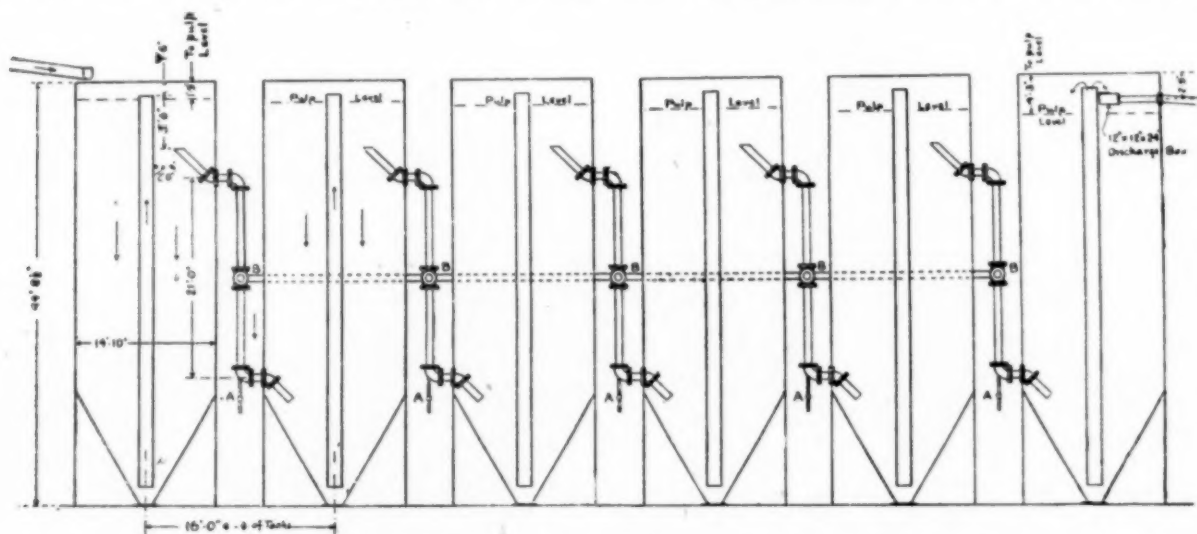
FIG. I.—THE CONTINUOUS SYSTEM OF AGITATION AT THE ESPERANZA MILL.

from the regrinding of the battery sands in 10 4-ft. x 20-ft. tube mills. It is with these "Lower Pachucas" this article deals.

Original Intermittent System.

Each tank held 80 tons dry slime and 144 tons of slime per 24 hours; the time required to fill one tank was 6.4 hours. After the available length of agitation each tank was discharged through a 5-in. valve at the bottom and pulp raised to the filter press storage tanks by means of three vertical bucket elevators. These elevators were made of an 18-in. 10-ply rubber elevator belt 54 in. C to C of head and boot pulleys, with 16-in. x 7-in. x 5½-in. type AA malleable iron buckets, spaced 18 in. C to C. The boots were made of cast iron and provided with accessible handholes for cleaning.

The housing, 2 ft. 6 in. x 5 ft. 6 in. inside dimensions, was built of No. 12 steel, with removable doors for repairs on belts, buckets, and pulleys. The original belt speed of 360 ft. per minute was raised to 540 ft. per minute to permit one elevator to be held in reserve. With the average condition of the elevators the time required to discharge one tank was about 3.8 hours.

On the assumption that one-half of the filling and discharging times resulted in net agitation on the entire tank charge, the net time lost in these operations amounted to 5.1 hours. The net agitation per charge, accordingly, worked out as follows:

*A paper presented on April 28, 1910, before the Instituto Mexicano de Minas y Metalurgia.

filter press storage tanks (collars of the storage tanks being 6 ft. below the collars of the Pachuca tanks and 50 ft. distant). The 6-in. pipe connections shown in Fig. 1 were put in Feb. 10, 1910, first as an experiment to demonstrate the applicability of the scheme. The particular information desired was with reference to comparative extraction and classification during agitation, as it was thought that the "sandy" product fed to the tanks might undergo considerable classification in passing from tank to tank. The temporary connections have proven so satisfactory that they have been retained in present daily operation.

The pulp is fed into the top of tank No. 1 at the rate of 300 tons dry slime and 540 tons solution per 24 hours and flows through the 6-in. pipes by gravity to the discharge box near the top of tank No. 6. The friction drop from tank to tank is about 6 in., making a total drop in the level of the pulp between tank No. 1 and tank No. 6 of 30 in. However, by utilizing the central tube as an air lift, and so placing the outlet into the final discharge pipe or launder that the overflow of the air lift will drop into the outlet, the difference in the level of the pulp in tank No. 1 and the level of the final discharge is reduced to 12 in. In fact, were it desirable to have the final discharge at a higher level than the first tank, the air lift in the last tank of the series could be raised sufficiently to permit of this being done (at an additional consumption of air, of course).

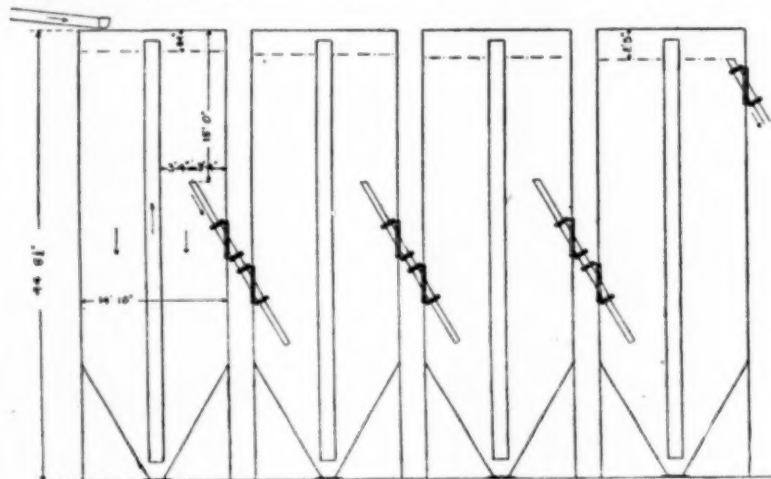The intake into a tank being near the bottom, the pulp rises

through the air lift, as the current in the annular space around the air lift is in a downward direction. The discharge from the tank is placed 7 ft. below the collar of the tank and 2 ft. from the side of the air lift, in this way "sampling" the overflow of the air lift and producing a practically uniform transfer of pulp from tank to tank, as shown by sizing tests and solid liquid ratios in the following table:
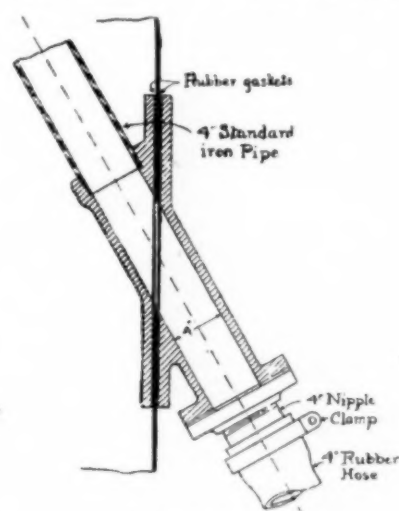
SIZING TEST OF PULP.

	Charge.	I.	II.	III.	IV.	V.	VI.
				Discharge from Tank No.			
+ 200 mesh	14.7%	14.9%	15.2%	15.7%	15.3%	15.1%	14.9%
— 200 mesh	85.3	85.1	84.8	84.3	84.7	84.9	85.1

Strength of K Cy solution in percents—

| | | .12 | .112. | .110 | .1075 | .105 | .1025 | .100 |

Consumption of K Cy per ton of ore, 358 grams.

Gold Contents and Percentage of Extraction.

	Charge.	I.	II.	III.	IV.	V.	VI.
+ 200 Mesh	9.7 gr.	46%	58%	61.5%	62%	63%	63.5%
— 200 Mesh	6.0 gr.	68	75.5	78	80	82	83.5
Gen. Sample	6.7 gr	66	72.5	74.5	76	77	78

Silver Contents and Percentage of Extraction.

	Charge.	I.	II.	III.	IV.	V.	VI.
+ 200 Mesh	75 gr.	9%	16%	22%	26%	28.5%	31%
— 200 Mesh	83 gr.	28	39	46	50.5	53	54
Gen. Sample	80 gr.	22.5	34.5	41.5	46	48	50

Feed and final discharge, solid to liquid ratio 1:1.80.

Sampling, six days at one-hour intervals.

Mr. Grothe thought that the mechanical part of the connections between the tanks, as shown in Fig. 1, could be improved upon, but the end discharge of the last tank was not open to objection. While the degree of sizing as reported by Mr. Kuryla is very satisfactory, the numerous curves in the pipes might be avoided. Especially the elbow at the discharge end of the connecting pipe is objectionable as it may become partially filled up by sediment, thus reducing the useful section of the pipe. The hydraulic grade between the successive tanks shows this to be the case. With a much smaller head the discharge through a 4-in. pipe should answer the purpose, if no obstruction existed. In the arrangement shown in Fig. 2 the possibility of clogging is entirely avoided. The pipes have an incline of 60 deg., and a moderate velocity of pulp keeps them clean. The connection outside the tanks consists of a flexible pipe, nullifying the effect of vibration and unequal expansion.

Mr. Grothe concludes that when the subject is better understood it will be found that continuous agitation is a great improvement on former practice. In fact, he can only see one

Section

Details of Constructions.

FIG. 2.—THE CONTINUOUS SYSTEM OF AGITATION AS SUPPLIED BY GROTHE AND CARTER.

The gain of extraction in the continuous system as compared to the intermittent is 1.3 per cent of gold and 1.5 per cent of silver and the saving in cyanide 25 grams per ton of ore.

The saving in operating labor and the increase in length of agitation effected in the "Lower Pachucas" has made it advisable to install the continuous system in the "Upper Pachucas" even though, as already mentioned, the "Upper Pachucas" discharge by gravity to the filter press storage tanks.

*   *   *

In the discussion which followed Mr. A. GROTHE referred to some experiments made by him and Mr. J. Leslie Mennell as to the effect of continuous agitation in laboratory tanks of 18-in. diameter and 54-in. height, the results being published in the *Mexican Mining Journal*, of February, 1909. He pointed out that Mr. Kuryla's paper is of particular value, as it removes the matter from the experimental to the practical stage. From his observations it is clear that an important saving in time, coupled with a decrease in cost and a slightly higher extraction, is obtained. Theoretically, the extraction in the continuous system should always fall a little short of the possible extraction in the intermittent and the reason that in Mr. Kuryla's case, as well as in Mr. Mennell's experiments, there was an increased extraction must be due to the fact that formerly the ore (in the intermittent system) was, for want of tankroom, not agitated as long as when the six tanks were connected and advantage was obtained in the latter case from the time formerly lost by filling and emptying, which now became available for some additional agitation. It seems certain from his observations that with the same number of tanks there is a decided advantage in favor of the continuous system.

disadvantage in the method (if it is one), viz., that a successful sampling of the pulp, on which the efficiency depends, can only be effected in "Pachuca" tanks.

Notes on Chemistry and Metallurgy in Great Britain.

(From Our Special Correspondent.)

The Iron and Steel Institute.

The spring meeting of the Iron and Steel Institute was held at the Institution of Civil Engineers on May 4 and 5.

The Duke of Devonshire's presidential address was mainly a remarkable comparison of international statistics of coal, iron and steel productions, and of populations during the last 40 years or so. (An abstract was given on page 314 of the June issue of this journal.)

First Day's Papers.

The papers by Mr. D. SELBY-BIGGE on "Development in the Production of Electric Power," its Application and Bearing upon the Iron and Steel Industries, and by Mr. E. G. SEHMER and Dr. R. DRAWE on economy and design of modern Reversing Rolling Mill Steam Engines were discussed together. Abstracts of these two papers will be found in the Synopsis in this issue.

In opening the discussion on Mr. Selby-Bigge's paper Mr. T. Westgarth said the utilization of waste heat was a matter of great importance to them as iron and steel makers. The existence of waste heat was self-evident, and the great point was to ascertain the cheapest way of using it. At Barrow a saving of

1500 tons of coal per week had been effected by putting in gas engines for blowing the furnaces and producing electricity for the auxiliary machinery; and the surplus gas was taken to the boilers to raise steam for the steel-works engines. At South Bank the waste steam was taken to regenerators and used in low-pressure turbines. Another firm sold their exhaust steam to a power company, who took it from the regenerators to low-pressure turbines. There were many ways of solving the problem of using waste energy, but it was not possible to lay down any general rule. Each case must be governed by its own conditions. Nevertheless there was a good deal to be said for the use of gas engines, because the surplus power from the coal after it had been used at the coke ovens and furnaces was probably more than could be obtained from the best engines and low-pressure turbines.

Mr. A. Lamberton said that it was not possible for him to attempt to discuss Mr. Selby-Bigge's paper in the short time allowed, but he fully recognized its great value. The paper by Mr. Sehmer and Dr. Drawe was also interesting, but it had some defects, and he would like to put some questions with a view of removing them. Some of the authors' figures seemed to institute comparisons between modern engines in Germany and a wasteful non-condensing engine referred to in a paper by Mr. Ablett last year. The authors stated that the Sehmer engine showed an economy of 35 per cent to 50 per cent in steam, but their figures showed that their engine used 350 lb. of steam as against 1100 lb. That was not 50 per cent. He made it 68 per cent. Such a discrepancy tended to throw doubt upon the whole question. He controverted the authors' statement that the introduction of low-pressure turbines between compound engines and the condenser was not a practical scheme, and quoted an instance showing an economy of 17 per cent in favor of the combination as against the reciprocating engine.

Mr. C. H. Merz agreed with Mr. Westgarth that the problem at every individual works must be solved in accordance with its own circumstances. Instead of employing exhaust-steam turbines or mixed-pressure turbines they might use gas engines, or put in an entirely new plant of high-pressure turbines. He had installed turbines at various stations totalling about 200,000 hp; he knew how most of them were operating, and he thought that in many cases exhaust turbines and mixed-pressure turbines were recommended where conditions were not favorable. He thought that in many cases exhaust and mixed-pressure turbines were put in in consequence of the reluctance of the manufacturer to scrap his old plant, which the exhaust turbines allowed him to retain, when a new turbine plant or installation of gas engines would have been better economy. The great thing in the use of exhaust and mixed-pressure turbines was the vacuum, without which they were practically useless, and the requisite vacuum was not always attained. He could not agree with Mr. Selby-Bigge's figures as to interest and depreciation, which would only allow them to write off their plant in 20 years. Turbines could be bought to-day giving 50 per cent better results than they did a few years ago. He thought that manufacturers should not jump to the conclusion that exhaust or mixed-pressure turbines gave additional power for nothing or that they were the best things, because in many cases they were not, and he would advise them to wait longer and completely reorganize their plant.

Mr. W. C. Mountain thought that, having regard to his experience of three collieries for several years, Mr. Selby-Bigge's figures as to cost per unit might be taken as correct. He had previously expressed his doubt as to electric winding being made commercially successful; and those remarks applied equally to electric rolling at present. The commercial aspect of these matters should be more fully considered. The lowest estimate a steam-produced horse-power could be put at was from $17.50 to $20 per annum.

Mr. W. J. Larke remarked that one question which had not been gone into was the use of the waste gas from blast fur-

naces and coke ovens to raise steam for high-pressure turbines. He knew that, in respect to thermal efficiency, this was not the best way of using the available gas, but the main point was commercial efficiency.

Second Day.

The discussion was resumed by Mr. Williams who said they had used gas from blast furnaces to raise steam for the steel works, and part of the exhaust steam was then used for generating their electric power and for blowing. The results appeared quite satisfactory, and they were going to join up another plant. They had found no difficulty in maintaining a vacuum of 28 in. to 28½ in., and, of course, below 28 in. the loss would be very great.

Mr. J. Mulholland advocated the use of high-power gas engines in iron and steel works. Their prime cost was no greater than that of steam plant, while repairs and maintenance cost less; and he thought that the superiority of the gas engine in thermal efficiency was generally acknowledged by engineers.

Mr. Selby-Bigge briefly replied. He agreed that in certain cases gas engines deserved to be considered as well as other means of producing power. Gas engines had recently been very much improved and were now very different from those produced even three or four years ago. He corroborated Mr. Lamberton as to the erroneous nature of the statement at the end of Mr. Sehmer's paper concerning the interpolation of a mixed-pressure turbine between the engine and the condenser. With regard to the question of difficulty in maintaining a vacuum he could support Mr. Williams' remarks. Personally, he has found that with mixed-pressure turbine plant 28½ in. was ordinarily obtained. They had got 29 in., but 28 in. was the most usual. Of course, it was necessary that the condenser should be big enough. The figures he had given for capital outlay were taken from an actual case.

Mr. Sehmer, in the course of his reply, said they did not wish to give a definite decision on reversing rolling mills, whether driven by steam or electric power, but only to mention some new features, the publication of which they considered desirable. The figures given by one of their newest engines showed that they effected a saving of from 60 per cent to 65 per cent.

Electric Furnaces.

Dr. W. Borchers' paper on "The Girod Furnace" (abstracted in the Synopsis in this issue) was then read. The discussion was opened by Mr. J. H. Heap, who said it was a reproach to us in this country that we had left the experimental work on the electric furnace chiefly to Frenchmen, although conditions afforded great opportunities to steel manufacturers for experiments with it. In the Middlesborough district they had waste gases affording as cheap a source of power as could be had anywhere. Even with the best hydraulic schemes the cost was as high, or higher, than that of using waste gases in an economical way. He was sorry Dr. Borchers had not given a general résumé of the electric furnaces at present in use, which, he thought, would have great interest for steel manufacturers who would be glad to have component and independent statements on the merits of the various types. He could not quite follow the author's reasoning in his approval of the Girod furnace. The only point which he could see indicating a special feature was in the contact pieces, and he doubted if the arrangement for cooling would enable them to last very long. The author seemed to assume that circulation of the contents of the bath would occur, but in some special arrangements that had not been the case. It seemed strange that it should have been left to Sheffield, which was by no means an ideal place, to take up the electric furnace. To obtain ideal conditions for the electric furnace they must have a cheap power and cheap material, and Sheffield producers had been obliged to have expensive raw material.

Mr. E. H. Saniter wished to ask Dr. Borchers if the Girod furnace consumed 1 kw-hour less current to produce a ton of steel than the Héroult furnace?. Dr. Borchers seemed to con-

sider that the Girod furnace was better because there was only one roof-hole and one electrode, and because—so the author stated—the resistance of the bath increased the heat there. Now the resistance of molten metal was the source of only a small proportion of the heat generated. In the Héroult furnace the reaction area was three times as great as in the Girod furnace, and he wished to ask the author if the smaller reaction area were an advantage.

Prof. J. O. Arnold said he supposed Mr. Heap would credit Sheffield with understanding its own business. Mr. Heap was repeating an old fable as to converting Middlesborough iron into the highest class of steel. That had been a menace to Sheffield for at least a century, but it had never been accomplished. What they had to consider was the relative melting capacities of the crucible and the electric furnace. They wanted a melting furnace, not a refining furnace, and therefore adopted the Kjellin furnace, and still used the finest Swedish iron, which produced the highest quality of steel known in Sheffield.

Mr. F. W. Harbord observed that he had had opportunities of seeing all the various types of electric furnaces except the Kjellin, and from an inspection of the Girod furnace last year he judged that it worked very satisfactorily, but he did not think Dr. Borchers should state that it was the simplest form of furnace. That depended on the particular purpose for which the furnace was designed. The advantage of one roof electrode instead of two did not apply to large furnaces, in which they must have two electrodes, and he believed in some cases they had three or four. With furnaces holding 200 tons of metal sufficient circulation was obtained without special means, and in small furnaces the ordinary reactions produced adequate circulation. He should expect trouble with the bottom electrodes because metal and slag were liable to collect round them when tilting, and the slag had to be splashed out. Repairs would also be troublesome.

Mr. D. F. Campbell said the figures in the paper did not warrant the statement that the Girod furnace was the simplest. Comparing a 12½-ton Girod furnace with a 15-ton furnace working at South Chicago, the Girod type had four roof holes and 16 bottom electrodes—and he thought trouble was more likely to arise with the bottom holes than with those in the roof—while the Héroult furnace had three roof holes and the bottom was arranged exactly as in the best basic open-hearth practice. The only sources of heat were the electricity and the mechanical reaction. He had no doubt there was ample circulation in any electric furnace, and had ascertained it in the Héroult type by adding ferro-chrome. In the Girod furnace the steel was liable to get under the layer of dolomite and hasten the breaking up of the lining. In South Chicago the bottoms of the Héroult furnaces lasted from 4000 to 5000 heats, and the cost of roof was about 4 cents per ton. When melting cold scrap the bottoms were protected by several hundredweight of lime and mill-scale, but in the Girod furnace the scrap was in contact with the pole pieces. The paper gave the time necessary for a heat with the Girod furnace as eight hours, whereas with cold scrap in a 2½-ton Héroult furnace the time of a heat was only slightly over six hours.

Carbides in Manganese Steel.

The next paper, by Dr. J. O. Arnold and Mr. A. A. Read, was on "The Chemical and Mechanical Relations of Iron Manganese and Carbon." (See abstract in Synopsis in this issue.)

Mr. J. E. Stead opened the discussion, and said he had always been of opinion that carbides of some kind—he would not say definite carbides—did diffuse in steel when cooled slowly, but the existence of a certain thing after cooling was not proof of the condition during solid solution. Dr. Campbell had, some years ago, suggested there might be different carbides in steel, just as there were different compounds of hydrogen and carbon, and this view seemed plausible. He (the speaker) had done some work in the same direction as Dr. Arnold's researches, and their conclusions were practically identical. He

would like Dr. Arnold to explain the different characters of pearlite represented in one of his photographs. Why was one granular, another lamellar, and a third described as troostitic?

Mr. W. H. Hatfield said the existence of a definite mixed carbide seemed to be established. According to the analysis a lower yield of carbide was accompanied by a higher percentage of carbon in the residue. He should like to have the result of similar experiments on supersaturated steel from which the carbide would come out without difficulty in larger masses.

Dr. Arnold replied that he could not explain the different phases of the pearlite. It was quite easy with the very fine lenses now produced for micrography to resolve so-called sorbite into laminæ. Besides the alloy was heated in a bar to a bright red for 36 hours, and then given three days to cool.

Tool Steel.

In the discussion on Mr. E. G. Herbert's paper on "The Cutting Properties of Tool Steel" (abstracted in the Synopsis in this issue), Mr. W. F. Beardshaw remarked that he hoped it would not be accepted that the author's method was the best possible one for ascertaining the relative values of tool steels, but judgment should be suspended until steel makers had had opportunities of testing the ingenious machine described.

Mr. King (Great Western Works) was of opinion that the conditions of test with the author's machine had little in common with those observed in machine shops in ordinary working. It seemed to him that the heat generated by the tool in the author's test was so small as not to compare even with that of a finishing cut in a tool shop.

Mr. A. Lamberton could not agree with Mr. King because it was shown that in some of the tests the tool was heated by gas jets to obtain workshop cutting conditions. The paper was of great value. The high prices asked for high-speed steel justified their asking the producers to specify the conditions under which it would give the best results.

Mr. Herbert, in his reply, said his objection to the usual working test was that it was carried out at one speed, and he had shown that variation of speed affected the result irregularly. They must test at all speeds to ascertain which was the best. In some cases a higher speed had increased the durability, just as his curve showed. Mr. King was under a misapprehension. The tests were made from low temperatures up to the highest the tool would stand.

Chromium Steel.

The paper on "Some Physical Aspects of 2 Per Cent Chromium Steels," by Andrew McWilliam and E. J. Barnes, was discussed with H. Moore's paper on "The A2 Point in Chromium Steel." Abstracts of these papers are given in the Synopsis in this issue.

Dr. W. Rosenhain questioned the correctness of Mr. Moore's differential heating curve. Indication of thermal change frequently occurred at a considerable interval from the temperature at which the bulk of the transformation took place. He considered that it had been shown that hardness was not maintained by quenching. It was really a case of the preservation of β iron by quenching, and the results Mr. Moore found in that case might naturally be expected.

Mr. E. H. Saniter asked Professor McWilliam if there were any specific reason for using so low a proportion of manganese in the steel. It seemed extraordinary that a temperature of 960° should be used for normalizing and annealing, while for quenching in water they had only 850°. If steel were heated to 1000°, allowed to cool, and then heated to AC1 and not to AC3, would there be any reorganization in its structure.

Professor McWilliam replied. He was surprised to hear Dr. Rosenhain express his belief in the existence of hard β iron. It was a question on which there was violent disagreement. How did Dr. Rosenhain define hard β iron?

Dr. Rosenhain, interposing, disclaimed having defined it. The reason for the hardness of hardened steel was the only practical point to be dealt with in connection with the hardness

of β iron. If it was only slightly harder than ordinary iron then it made no appreciable difference. Everyone knew that was the point at issue. In answer to Mr. Saniter, there were both practical and scientific reasons for keeping the manganese low, because they wanted to see the effect of chromium. Mr. Moore also briefly replied.

Iron and Steel Production in the Transvaal.

The Transvaal Mines Department has issued a report by Mr. F. W. Harbord on the prospect of manufacturing iron and steel in the Transvaal. Although there are good deposits of magnetic quartzites near Pretoria and at Airlie, which, if mixed with small quantities of richer non-siliceous ores existing in other localities, could be smelted to produce a pig iron suitable for steel making by the basic process, yet the supply of suitable limestone and coke, except at excessive cost, makes it improbable that the manufacture of iron and steel in the Transvaal is likely to be a commercial success. Labor does not present any insuperable difficulty, but the question of a market for the finished product deserves serious consideration.

A steel company in the Transvaal could hardly hope to secure more than 50 per cent of total imports to South Africa, which would mean about 50,000 tons per annum of all classes of materials.

If all the railways of South Africa would combine to take a fixed fairly large quantity of material per annum, the situation would be more promising, and an iron and steel works turning out 1000 tons of finished steel per week should then be able to live.

Mr. Harbord comes to the conclusion that any attempt to start the manufacture of pig iron and steel on a large scale would at present be premature; but, on the other hand, electric steel plant to deal with the large quantitiees of scrap at Pretoria and other places in the colony promises good results, both technically and commercially. After careful consideration of all local conditions, both as to quality of scrap and finished products required, he is of opinion that an electric steel furnace is by far the most suitable and will give the best results in actual practice.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Electric Power for Iron and Steel Works.—A paper presented by Mr. D. Selby Bigge at the recent meeting of the Iron and Steel Institute deals with "developments in the production of electric power; its application and bearing upon the iron and steel industries." He deals especially with the progress that has been made in the cost at which electricity can now be produced in iron and steel works having at their disposal waste gas, waste heat, and waste steam. One of the means by which a lower cost of production has been attained has been brought about by the advent of the mixed-pressure turbine. The author discusses at some length mixed-pressure turbines and then refers to the fact that the introduction of the steam turbine has created a demand for cooling plant in cases where a natural supply of cooling water is not available, and where a good case exists for the introduction of mixed-pressure or exhaust-steam turbine plant. There are several types of cooling plant on the market giving excellent results. The three forms most commonly met with are the natural-draught cooling tower, the open-type cooling tower, and spray nozzles distributed over a pond. In England the circulating water can be reduced to a temperature of 80° Fahr. to 85° Fahr. under ordinary conditions with a natural-draught cooling tower. The open-type cooling tower will give better results, and circulating water temperatures of 75° Fahr. may be obtained with its use if situated in a favorable position. This type of tower, however, requires more ground space than the natural-draught type. Where ground space is

not of great importance, and especially in open situations, spray nozzles are to be recommended, as excellent results are obtained by their use, and if properly installed, it is quite easy to maintain the temperature of the circulating water at the prevailing atmospheric temperature. Details are given about the following types of mixed-pressure turbines: Parsons, Ratean, Curtis, Bergmann, Dr. Laval, Zoelly, Westinghouse. Recent development in gas engines are then taken up (Mather & Platt, Nurnberg, Cockerill-Westgarth).

In discussing the commercial aspects of electric power schemes with the aid of waste heat, the author gives the following details of the electrification scheme, which has recently been carried out by the Fife Coal Company in connection with their collieries in Scotland. In this scheme the principle adopted has been to lay down mixed-pressure turbines at those collieries where the greatest amount of exhaust steam might be available, and feed the power thus generated into a common network of mains, from which power could be drawn off to supply the requirements of their other collieries in the district. The Fife Coal Company possess five mixed-pressure turbines, each of 1000 hp and one of 800 hp. The current is generated at 3000 volts at a frequency of 50 cycles. The turbines run at a speed of 3000 r.p.m., and in addition to the direct-coupled exciters upon the shaft, separately driven steam excitation sets are provided for emergencies. Each plant is complete with condensing and water-cooling arrangements. It is yet too early to gauge the total savings resulting from this comprehensive electrifica-

tion scheme, but at one of the collieries, where a 750-kw plant has now been in operation for a year, the cost of production, after allowing 12 per cent interest and depreciation upon the capital outlay, including a complete standby unit, and including superintendent's wages, oil, stores, waste, etc., and an allowance for live steam, amounts to 0.4218 cent per kw-hour, made up as follows:

Interest and depreciation at 12 per cent per year upon cost of complete station equipment, including standby unit ($47,000) $5,640
Station attendants 800
Stores, oil, etc.. 225
Coal for live steam and firemen's wages................ 3,420
_____
$10,085

Total load ....2,295,000 kw-hours.
Cost of production ....0.4218 cent per kw-hour.

At this particular colliery, the net saving which has been actually effected already exceeds $15,000 per year, but as the plant is not yet fully loaded up and more motors have to be added, the ultimate saving will be considerably increased. Brief notes are added on electric rolling mills, electric furnaces, transmission lines and voltages of installation. (From advance sheets.)

Rolling-Mill Steam Engines.—An Iron and Steel Institute paper by Mr. EDUARD G. SEHMER and Dr. R. DRAWE deals with the "economy and design of modern reversing rolling-mill steam engines." They believe that the electric drive of rolling mills, electricity being generated from gas power, is a mistake. They describe a simple single-lever gear for steam engines for rolling mills so designed that the steam consumption is a minimum. The output of the work of the engine is controlled by regulation of the cut-off, the proper amount of cushioning takes place, and the number of idle revolutions is very small. Reference is made to a 45-hour test on a rolling mill compound

FIG. I.—VERTICAL SECTION OF GIROD FURNACE.

engine recently carried out by the Commission on the Power Requirements of Rolling Mills. The mean pressure on the engine was 103 lb.; the mean temperature vacuum was 23.5 in. The average steam consumption was 350.77 lb. per ton of material rolled down to 9.222 times its original length. Under equal conditions, and with the same amount of elongation, the steam consumption was formerly 880 lb. to 1100 lb. per ton. The

amount of steam consumed includes the whole of the feed water necessary for the engine, together with all the auxiliary engine and steam pumps for the condenser.

The authors believe that the advantages of steam turbines are often overrated and refer to experiments carried out at the "Union" works at Dortmund on engines whose exhaust steam was applied to driving rolling mills. The result showed that the steam consumption of a compound rolling-mill engine—
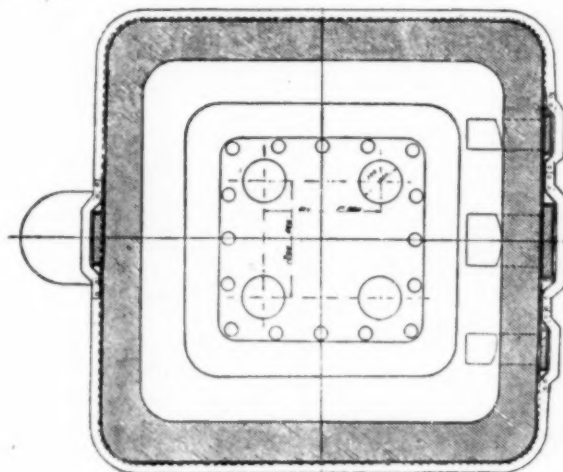
FIG. 2.—PLAN OF GIROD FURNACE.

that is, a very economically working one—was increased by 35 per cent by the interpolation of an exhaust-steam turbine between it and the central condenser. The experiments led to the conclusion that no economical advantage results from the use of such a turbine, for the chief reason that the extra 35 per cent of steam used in the reversing engine would be sufficient to develop the same amount of energy in a live-steam turbine as would be obtained by means of installing the rather complicated and costly exhaust-steam turbine plant. (From advance sheets.)

Electric Steel Furnace.—Prof. W. BORCHERS has written another paper on "The Girod Furnace," this time for the Iron and Steel Institute. He emphasizes again his opinion that "the Girod furnace is the simplest of modern electric steel furnaces, both as an experimental furnace, or as a smelting works furnace on the largest scale possible. In the author's laboratory it was being used with current of 30 kw to 36 kw, while elsewhere, for steel smelting, it is being used with currents of 300 kw to 1200 kw." Figs. 1, 2 and 3 are sections of a 12-ton Girod furnace. There are four upper electrodes and 16 "contact pieces" in the bottom. The furnace requires 1000 kw to 1200 kw at 70 volts to 75 volts. Taking into account a loss of 10 per cent to 11 per cent of the charge by oxidation and evaporation, the power required to make one ton of steel in this furnace is 800 kw-hours to 900 kw-hours starting with cold charges. The consumption of carbon electrodes amounts to 12 kg or 15 kg per ton of steel, including the short pieces, which remain in the holders when the latter have reached the lowest point at which they can withstand the radiating heat of the bath. The lining of the furnaces, mostly consisting of calcined dolomite, will withstand at least 80 charges. Thereafter the side walls, into which cavities have been eaten by the slag around the level of the molten bath, will require repairing, and these repairs will subsequently need to be effected at shorter intervals. The bottom of the furnace will, however, stand 120 to 150 charges without repair. The thickness of the bottom will by that time have been reduced by about 100 mm. Furnace linings will stand a considerably longer time if liquid charges are used. One furnace working in this manner was run for 200 charges without any repairs. The linings of the furnace covers require more frequent renewal. In the small furnaces they need repairs after 25 to 30 charges, and in the
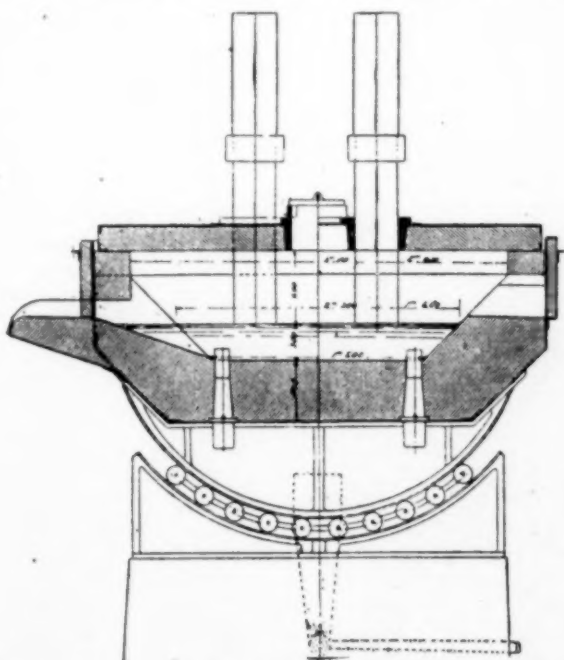
larger furnaces after 20 to 25 charges. The other parts of the furnace are not worn out as fast as the linings and electrodes. The furnaces which have so far been put into operation still possess their original outfit. At the works at Ugine only scrap iron is used as raw material for steel making as well as for the manufacture of iron alloys. The larger part of the scrap is soft iron and steel, the smaller part pig and cast iron. Some notes are added on the equipment of the works
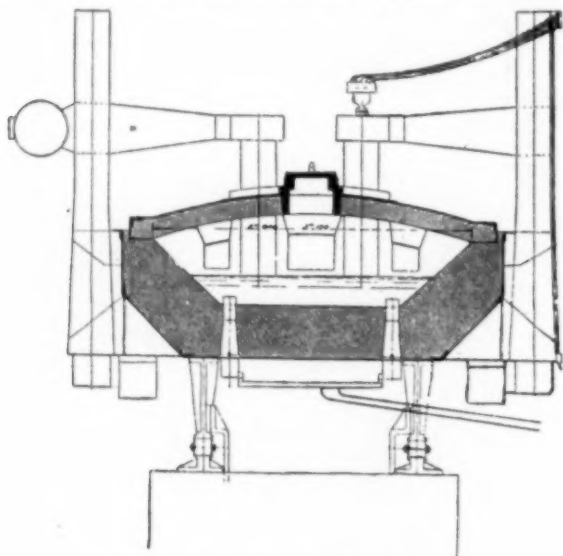

FIG. 3.—VERTICAL SECTION OF GIROD FURNACE.

of the Society Anonyme Electrometallurgique, Paul Girod. (From advance sheets.)

Carbides in Manganese Steel.—An Iron and Steel Institute paper by Prof. J. O. Arnold and Prof. A. A. Read deals with "the chemical and mechanical relations of iron, manganese and carbon." The authors made a number of experiments to determine the composition of the carbides separated from a series of annealed steels containing various percentages of manganese, the percentage of carbon being practically the same in each. The alloys were made as nearly as possible to contain about 0.9 per cent of carbon with ascending manganese. The color of the carbides obtained by separation ranged from bright silvery gray in the lowest to dull gray in the steel with the highest percentage of manganese. The percentage of carbon in the carbide was found in most cases somewhat higher than the theoretical amount 6.7 and also, in some cases, the total amount of carbon in the steel was not obtained as carbide. This is only to be expected when dealing with a substance like manganese carbide, which easily decomposes.

As the manganese rises so does the amount of iron replaced by manganese in the carbide go on increasing until a steel with 4.98 per cent of manganese is reached. From this point to a steel with 13.38 per cent of manganese, the manganese in the carbide remains stationary. With steel containing 15 per cent of manganese, the manganese in the carbide begins to increase again, and probably with a 20 per cent manganese steel a further replacement of iron by manganese takes place. It is somewhat difficult to say whether the double carbides are true compounds or simple mixtures of iron and manganese carbides. It would appear that the carbides from the steels up to 4.98 per cent of manganese are simply mixtures, and possibly a considerable amount of a true double carbide, $3Fe_3C, Mn_3C$ is present in steels containing 11.21 per cent and 13.38 per cent of manganese respectively. In steels with 15 per cent of manganese and upwards, a double carbide containing a higher percentage of manganese, such as $2Fe_3C, Mn_3C$, or a mixture of carbides, may be formed, a point being reached resulting in the formation of another true double carbide. A mixture of carbide of iron and manganese, or a double carbide of iron and

manganese, registers a lower percentage of carbon by the color test than carbide of iron per se. With steels up to 10.07 per cent manganese, the percentage of total carbon agrees very closely with the amount of carbide of iron present in the carbide, thus showing that manganese carbide gives no coloration with HNO_3, specific gravity 1.20. The increase of total carbon obtained by the color test with steels containing 11.21 per cent and 13.38 per cent manganese seems to point to the formation of a compound capable of giving a slightly higher color. Results of micrographic analysis are also given. (From advance sheets.)

Tool Steels.—An Iron and Steel Institute paper by Mr. Edward G. Herbert deals with "the cutting properties of tool steels." Tool steels are steels which can be made hard enough to cut metals. But hardness alone does not determine the quality of tool steel, since a tool should not only cut, but continue cutting for a considerable time before becoming blunt. The author has devised a tool-steel testing machine, the principle of which is shown in Fig. 4, where A represents the element on which the cutting tests are made; that is, a tube, usually of steel, ¾ in. in diameter and ⅝ in. bore, formed by drilling from the solid bar, and subjected to a heat treatment which renders it uniform in hardness. The tube A revolves about a vertical axis, while its lower end rests on a fixed abutment B of hardened steel. The sample of steel to be tested is made into a cutting tool C, which is ground to a standard angle and held in a vise D with its cutting edge radial to the tube. The vise D is mounted on knife edges E lying in the plane of the end of the tube. By means of a steelyard F, with a movable weight and scale, the tool can be caused to press upward against the edge of the tube with any desired force, and cut it away with a turning action. The tube is held in contact with the abutment B by means of a weight G, the downward pressure of which is always greater than the upward pressure of the tool. As the tube is turned away by the tool, it is fed downward by the weight G, so that the end of the tube, though it is constantly being turned away, is constantly held in contact with the abutment B and the point at which cutting takes place is stationary. A dashpot H is connected to the vise to prevent jarring or vibration. As the test proceeds, the cutting edge is gradually worn away, and as this takes place the vise tilts on the knife edges. The angle through which the vise tilts is proportional to the blunting of the tool, and this is actually measured in thousandths of an inch by means of the micrometer K, which is mounted on the pillar of the machine, and is brought to bear on a projection from the end of the steelyard, so as to measure its movements due to the wear of the tool.

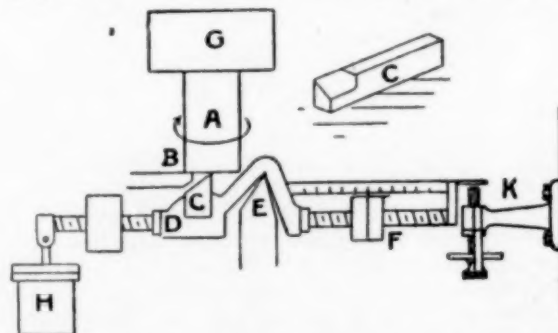Fig. 5 gives a view of the tool-testing machine. The tube

FIG. 4.—DIAGRAM OF TOOL-STEEL TESTING MACHINE.

is driven by a spindle, which receives its motion through cone and friction disk gearing, capable of giving it a wide and continuous range of speeds. A paper-covered drum, mounted beside the column of the machine, is driven through worm-and-spur speed-reducing gearing from the spindle. Vertical movements of the spindle are communicated through a fine chain to a pencil mounted on a vertical sliding bar. When the ma-

chine is set in motion the pencil draws a diagram, in which horizontal distances represent the number of revolutions made by the tube, and vertical distances represent vertical movements of the spindle—i.e., the length of tube turned away by the tool. The slope of the line represents the rate at which the tube is being turned away, or, in other words, the feed or traverse of the tool in inches per 1000 revolutions of the tube. By means of this instrument tests can be made at varying speeds up to the highest speed at which the tool will cut, and the results plotted in form of "speed curves" in which the ordinates represent the durability of the tool and the abscissas represent cutting speeds.

A set of speed curves is shown in Fig. 6, in which *A* is the curve of a carbon steel. *B, B'* are both from the same high-speed steel differently hardened, and *C* is from another high-
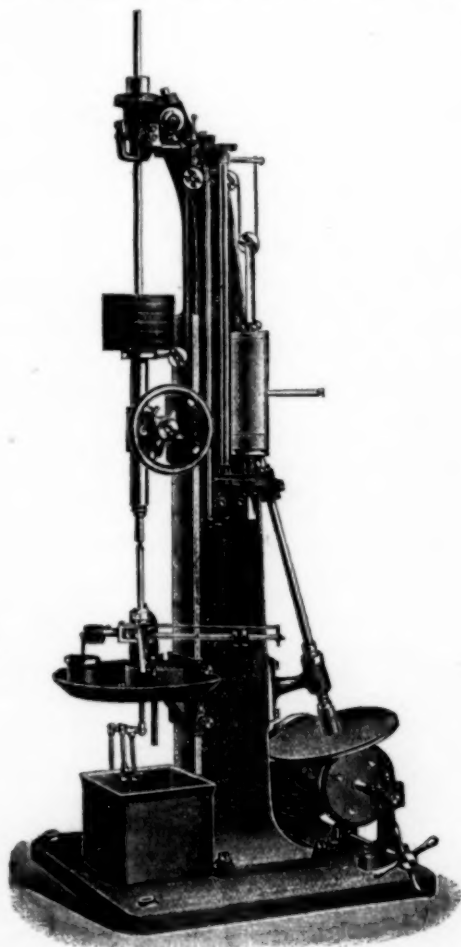
FIG. 5.—TOOL STEEL TESTING MACHINE.

speed steel. At the low speeds, up to 50 ft. per minute, the carbon steel is more durable than the high-speed steels; but at the higher speeds the carbon steel is (as might be expected) less durable. The fact that a high-speed steel tool will not keep its edge so well as a carbon-steel tool under a light cut at a low speed (a finishing cut) has long been familiar to engineers, but the fact that the durability of a high-speed steel tool working under a light cut, can be greatly increased by running it at a high speed, and that under these conditions—a light cut and a high speed—it will keep its sharp edge better than a carbon-steel tool, was not generally known. Perhaps the most striking and unexpected feature of the speed curve is this: that the durability of all steels without the exception is very low at low speeds (under light cuts) and that the durability of the cutting edge increases as the speed is raised. In considering these curious facts, it is important to bear in mind that we are here dealing with durability as meas-

ured, not by the length of time the tool will continue cutting before becoming blunt, but by the amount of metal it will cut away before becoming blunt. To turn away an inch of tube at 40 ft. per minute obviously takes twice as long as turning away an inch at 70 ft. per minute cutting speed, the thickness of the shaving being the same in each case. If the speed curves are drawn out on a time basis they assume a rather different form, as shown in Fig. 7. From this it appears that the maximum time durability of the carbon steel was actually higher than that of the best high-speed steel, but that this maximum occurred at a low speed. Neither the curves in Fig. 6 nor those in Fig. 7 give a correct view of the relative value of carbon and high-speed steels for the rapid removal of metal. The best steel for this purpose is not the one that will keep its edge longest when run at a slow speed, nor even the one that will remove the greatest amount of metal before becoming

FIG. 6.—SPEED CURVES. (A, CARBON STEEL; B, HIGH-SPEED STEEL; B', SAME TOOL REHARDENED; C, HIGH SPEED).

blunt. The engineer usually requires the steel that will remove the greatest amount of metal per hour without requiring too frequent sharpening. If we multiply the number of inches of tube turned away by the corresponding cutting speed, we obtain a quantity which may be termed the "duty" and which is proportional to the time rate of removing the metal, and to the durability of the tool. It is reasonable to assume that when this quantity attains a maximum the ideal working conditions have been reached. In Fig. 8 are shown duty curves obtained from the speed curves, Fig. 6, and this diagram shows the relative values of the steels for the rapid removal of the metal. Fig. 9 gives an even more striking comparison between the duty curves of A, a carbon steel; B, a good high-speed steel, and D one of the new vanadium steels. When the vanadium steels were first brought out it was generally supposed that they were not capable of working at higher speeds than the older tungsten steels. The tool-steel testing machine has shown that they are capable of working, and that they may attain their maximum

FIG. 7.—SPEED CURVES ON TIME-DURABILITY BASIS.

durability at speeds which are much too high for the ordinary tungsten steels, and this has now been amply confirmed in practice.

The author then discusses at some length the causes of the differences in the durability of the cutting edge of a given
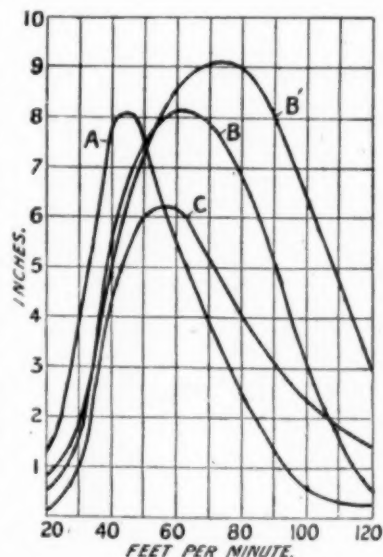
steel at different speeds. The durability of the cutting edge cannot depend on the rubbing of metal against metal alone. It seems that it depends also to a large extent on the evolution of heat due to the rubbing and causing a rise in temperature of the cutting edge. It seems that each speed on the testing machine corresp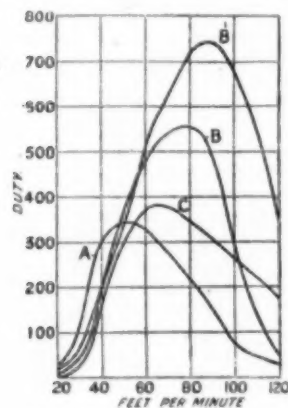onds to a definite temperature of the cutting edge and that the durability of a tool (of a given shape and cutting a given material) depends entirely on its temperature. From this point of view the speed curve is really a temperature curve. While the above curves are simple in form, more complicated curves often appear, showing two peaks with a distinct depression between the two. This corresponds to an almost total loss of durability at a particular speed which may be intermediate between two speeds, each of which gives the tool the highest durability of which it is capable.

FIG. 8.—DUTY CURVES.

Unaccountable failures of cutting tools are among the commonest of workshop experiences and may be frequently due to working at the critical temperature of low durability. The concluding chapters of the very long paper deal with the effect of temper on the durability of carbon steels and the effect of percentage of carbon on durability, and finally the effect of the cooling process on the durability of high-speed steels. (From advance sheets.)

Chromium Steels.—An Iron and Steel Institute paper by Prof. A. McWilliam and Mr. Ernest J. Barnes deals with "some physical properties of 2 per cent chromium steels." Tensile tests, the effect of heat treatment, heating and cooling curves, and micro-photographs are described and discussed. Another Iron and Steel Institute paper by Mr. Harold Moore deals with "the A-2 point in chromium steel." The following conclusions are reached: The position of Ac1 is progressively raised by additions of chromium to steel, and if sufficient chromium (3 per cent) is present, another critical point occuring at about 777° C. and below Ac1 is indicated by heating curves. The identity of this lower critical point with the magnetic change point proves it to be Ac2. The occurrence of Ac2 below Ac1 is held to support the view that Fe3C is insoluble in β-iron. The method of determining the magnetic change point, described above, is simple and accurate. (From advance sheets.)

An account of the discussion which followed the different papers, abstracted above, at the recent meeting of the Iron and Steel Institute, will be found in the letter of our London correspondent in this issue.

Alleged Lack of Uniformity of Electric Steel.—In this journal, January, 1910, Grönwall had claimed that in many cases in practice there was found lack of homogeneity and

FIG. 9.—DUTY CURVES. .(A, CARBON STEEL; B', HIGH-SPEED STEEL; D, THE NEW STEEL).

uniformity of steel produced in electric furnaces from cold charges. This was claimed to be due to differences of temperatures within the furnace. In *Stahl und Eisen* of April 20, V. Engelhardt replies to this attack as far as the induction furnace is concerned and gives some interesting notes on the automatic motions which are always observed in induction furnaces. They are indicated in the adjoining diagram, in which 1 represents a simple induction furnace, 2 a Roechling-Rodenhauser furnace operated with single-phase current and 3 a Roechling-Rodenhauser furnace operated with three-phase currents. In all of them there are two different kinds of motions. First, the rolling motion perpendicular to the surface of the steel bath and directed toward the inner wall of
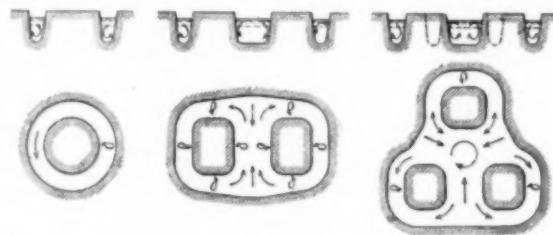

FIG. 10.—MOVEMENT OF CHARGE IN INDUCTION FURNACE.

the crucible. Second, a rotary motion parallel to the surface of the bath, at a lower speed. The motion is not so strong that the steel and slag would get mixed up, but is strong enough to produce absolute uniformity of the steel. This has also been confirmed in actual practice.

Titanium in Steel.—An article by W. Venator in *Stahl und Eisen* of April 20 gives the results obtained in practice in a steel works in Osnabrück, in Germany, with the use of titanium in the manufacture of basic open-hearth steel. The molten steel was divided into two halves, to one of which titanium was added, the additions varying between 0.038 per cent and 0.14 per cent. Detailed comparative tests were then made between the steel treated with titanium and the untreated steel. The ingots of the titanium-treated steel were found to be perfectly sound in rolling. The chemical analysis showed that the titanium was present only in traces in the steel, it having passed chiefly into the slag. The mechanical tests showed conclusively that the treatment with titanium is advantageous, as it toughens the steel; it also has an effect on the microstructure, as is shown by the appearance of the fracture. The results of the mechanical tests are given in detail. The author thinks that the success of the titanium treatment is mainly due to very effective dioxidation. That the titanium also removes nitrogen from steel needs to be proven. The titanium acts simply as a purifying, not as an alloying, agent. It is stated that in a crucible steel works favorable results were obtained by adding metallic titanium mixed with aluminium file to the steel before pouring. It was found that the aluminium had no disadvantageous effect on the steel. It apparently facilitates the diffusion of the titanium.

Gold and Silver.

Chlorine in the Cyanidation of Silver Ores.—The development of the electrolytic cell for the cheap production of chlorine is giving rise to a number of investigations looking toward the use of this element in the extraction of the precious metals. It is well known that chloridizing roasts can be utilized for treating refractory silver ores, but recent investigators tend more and more to adopt more direct methods of treatment. Writing in the *Pacific Miner* for May, 1910, D. Mosher suggests a scheme for chloridizing silver ores containing antimony and other base combinations by the direct application of chlorine in a chlorine-proof tank, subsequently extracting the silver with cyanide. His idea is to first slime the ore and treat the pulp in the ratio of one of slime to one of water heated to 120° Fahr. to 150° Fahr., by passing chlorine into the sludge until tests show the conversion of the sulphides into
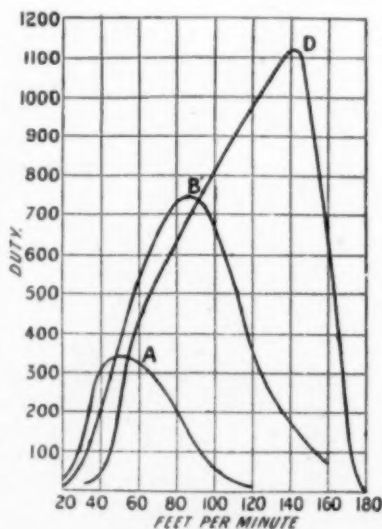
chlorides. Caustic soda formed in the cell which produces the chlorine may be used to neutralize any acid salts formed in the reaction. After dilution and subsequent thickening of the chloridized slime, silver and gold are to be extracted by cyanide in any customary manner. The construction of the chlorine-proof tank is suggested as follows: An outer steel shell lined with reinforced concrete, attached to which is a double layer of acid-proof brick set in acid-proof cement.

Cyanide Practice in Western Australia.—An outline of the treatment of concentrates at five mills operating on the "Golden Mile," Kalgoorlie, Western Australia, is given by W. M. Von Bernewitz in the May 21, 1910 issue of the *Mining Journal* (London). Wet crushing, concentrating, roasting and cyanidation are the main features of the process. Edwards Simplex and Duplex roasters are used at four of the five mills, the other using the Merton Improved. At all of the plants except one a sweet roast is made. At the other, 4 lb. to 5 lb. of salt are added to the roaster charge and a poorer roast is made. Following the roasting process the product is treated in grinding pans so that about 80 per cent passes a 200-mesh screen. Agitation with weak cyanide solution, filtration and washing in filter presses, and precipitation in zinc boxes complete the steps in the process. The amount of flue dust formed is stated to be very low, one plant getting about 0.4 per cent of the raw tonnage roasted. The cyanide solutions in which the slime is agitated vary in strength from 0.03 to 0.1 per cent KCN. Air agitation is used for about 12 to 36 hours. Filtration is carried on under 75 lb. to 100 lb. pressure, and washing at from 50 lb. to 90 lb.

Chlorination and Charcoal Precipitation.—An interesting description of the Edwards Pyrites Works, Bendigo, Victoria, Australia, is given by Donald Clark in the *Australian Mining Standard* for April 27, 1910. Sulphide concentrates are roasted in Edwards Duplex and tilting furnaces. The capacity of the Duplex roaster is given as about 14 tons of clean pyritic concentrates in 24 hours, using about 4.5 tons of wood. The cooled roasted ore is mixed with about 10 per cent water and charged into chlorination vats. Chlorine is forced in at the base of the vats under slight pressure and allowed to flow as long as it is absorbed. From 24 to 48 hours are required for complete solution of the gold as chloride, after which water is turned into the vats at the top and allowed to carry the gold chloride in solution through the customary gravel-sand filter. The gold solution is pumped to suitable storage vats, whence it flows through the precipitating jars. The latter are in three series arranged in steps. Each jar is 3 ft. in height with a diameter of 1.5 ft. An earthenware pipe is set vertically in the center of each jar, and the space between this and the sides of the jar is packed with lumps of clean charcoal. The pipe may also be similarly packed. The gold solution is made to flow at the rate of about 3 liters per minute downward through the central pipe, upward through the charcoal outside and thence to the next jar. After passing the series of three jars the solution is conducted through two large charcoal-filled vats before it is allowed to run to waste. The precipitation is said to be remarkably efficient, and 90 per cent of the gold will be precipitated in the first series of jars. The charcoal is slowly burned and the ash melted for recovery of the gold. Chlorine is generated from manganese dioxide, salt and sulphuric acid.

Copper.

Bessemerizing Copper Matte and Copper-Nickel Matte.— A study of the behavior of copper-nickel mattes in the Bessemer converter, extending over a period of five years, is contributed to the April *Bulletin* of the American Institute of Mining Engineers by David H. Browne, metallurgist of the Canadian Copper Company. Nickel was formerly regarded as an element replacing iron in copper matte, but as a result of an extensive investigation the author announces certain conclusions, as follows:

1. Nickel is not an element replacing iron in matte.
2. Nickel-copper alloys act in the matte-blow like one metal.

3. Nickel-copper alloys follow, in the matte-blow, exactly the same laws as govern the behavior of copper alone.

In arriving at the laws governing the behavior of copper during the blow, and in determining the relations of copper to sulphur and iron, the author makes use of data previously contributed by other investigators, notably Mathewson. By recalculating Mathewson's analyses showing the "rate of elimination" of matte impurities, to express "ratios" of iron and sulphur to copper present in matte at a given time, the author shows that the curve of iron elimination is a regular mathematical curve, and that, on the whole, sulphur disappears in a straight line, with a curious jog or reversal when the iron is down to 5 per cent. The conclusion is, therefore, reached that "relative elimination" is a fixed factor, no matter how much the "rate of elimination," or blowing time may vary. This suggests the idea of the laws governing the action of copper matte in the Bessemer converter. Experiments at the plant of the Canadian Copper Company show that in bessemerizing copper-nickel matte, nickel does not follow the iron into the slag; that only in the last stages of the blow does it oxidize; that when it does pass into the slag it carries more than its own weight of copper with it, and that it is impossible to blow all of the nickel out of the matte. The elimination of the sulphur and iron is, therefore, a direct function of copper-nickel in such mattes, and not of either metal separately.

Vacuum Concentration.—The application of the Elmore process to hitherto wasted tailings at Sulitelma, Norway, within the Arctic Circle, is an interesting development resulting from experiments begun several years ago. H. Holmsen and H. N. Rees describe the plant in the May *Mining Magazine.* At present a plant of 12 Elmore vacuum concentrators is treating 500 tons of tailings per day, producing 1000 tons of concentrates per month. The annual production of metallic copper from this plant is from 800 tons to 1000 tons. The vacuum plant (Fig. 11) treats tailings from jugs and concentrating tables in which there was formerly a loss of about 40 per cent copper. This material is delivered to a large spitzkasten, the coarse underflow from which is reground in Krupp ball mills, the overflow going to settling tanks. The overflow from the latter goes to waste. The crushed product from the ball mills, the fine from the spitzkasten and the slime from the settling tanks are delivered to a rotary distributor which accurately divides the pulp into equal parts for the separate units of the Elmore plant. The consumption of oil is small, amounting to
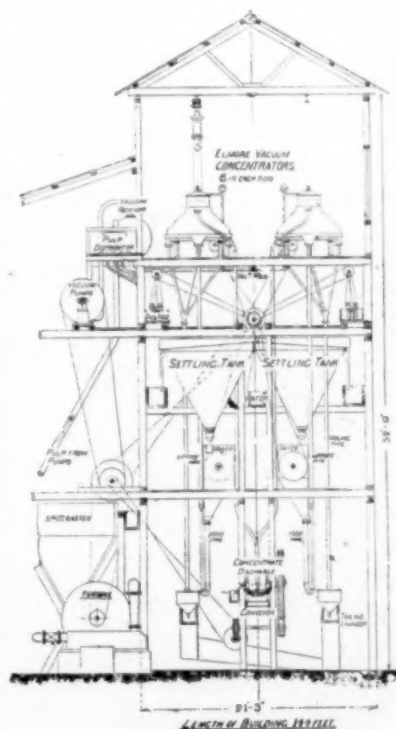

FIG. 11.—VACUUM PLANT AT SULITELMA.

2.7 lb. per ton of material treated. No acid is used. Formerly sulphuric acid was used at the rate of 3 lb. per ton of material treated, and later experiments with sulphurous acid prepared by passing roaster gases through water also gave satisfactory results. Neither is used now, although slightly better results might be obtained by the use of acid.

The following figures indicate the efficiency of the process without the use of acid:

	Per Cent Copper.
Tailing from jigs and tables	0.8 to 1.4
Concentrates	6.0 to 8.5
Final residue	0.15 to 0.3
Extraction	Over 80 per cent.

The labor employed is one man for each floor per shift of eight hours, viz., at the separators on the top floor, at the mixers on the first floor and at the concentrate discharge on the ground floor. The plant has had constant operation, losing but 25 hours in eight months.

Converting High-Grade Mattes.—In the March 5, 1910, issue of the *Aust. Min. & Eng. Rev.*, W. A. Longbottom describes the successful bessemerizing of matte carrying 66 per cent to 69 per cent copper. Owing to a shortage of suitable low-grade material the furnaces of the Electrolytic Refining & Smelting Company become filled with high-grade matte, and although the copper content of the material was far higher than was considered advisable for treatment in the Bessemer converter, it was decided to attempt the experiment. The converters are 10 ft. 6 in. by 7 ft. 6 in., and the air pressure available was only 9 lb. per square inch. The greatest trouble experienced was in keeping the tuyeres clear, and a man was kept constantly punching to accomplish this purpose. The slag was very viscous and at the time of skimming it was necessary to rabble the slag off from the converter. The linings withstood the operation much better than with the more fiery low-grade matte. The time required to blow the slag averaged from 50 to 60 minutes, but the time required to blow to blister was relatively long, being as high as 2 hours and 50 minutes to 3 hours. The best results were obtained with a charge of about 6 tons of matte, and it was found that if this quantity was much increased the time required to blow to blister was greatly increased, probably on account of the low air pressure and the loss of tuyeres due to slagging. The author recommends the alternate treatment of high and low-grade mattes in the converter, rather than mixing two mattes of extreme grade. An experiment of the latter kind gave poor results.

Vortex Copper Converter.—In the ordinary form of copper converter the air is introduced at the angle of greatest resistance and the force of the air current is spent largely in overcoming frictional resistance due to impact in encountering the wall of liquid matte. Thus the power stored up in the many air currents is not used to great advantage. In the May 7, 1910 issue of the *Eng. & Min. Journ.*, Herbert Haas describes his suggestion for a vortex converter, in which the power of the air currents is used to set up a rotation of the converter contents and thereby insure a more thorough mixing of the various constituents and a more thorough and rapid oxidation of the iron and sulphur. He uses a spherical converter blown from the bottom. The tuyeres are so located and designed that the air currents are directed forward, outward and upward, thus following the line of least resistance and imparting a motion to the mass. This would cause even corrosion of the lining, eliminate the necessity of punching the tuyeres and prevent the formation of noses. A still further advantage would lie in the thorough mixing of the constituents Cu_2S and Cu_2O, with consequent reduction of copper. Owing to the difference in specific gravity of these constituents they tend normally to separate. The vortex converter has never been placed in operation and is offered as a suggested improvement.

Electrolytic Refining.—In five issues of the *Aust. Min. Stand.*, from Feb. 16 to March 16, 1910, inclusive, H. Schroder describes in detail the Australian practice in electrolytic copper refining as carried out at Wallaroo, Lithgow and Port Kembla. The author has been officially connected with each of the three plants. Only a few points can be selected from the details given in the series of articles. At Lithgow the tanks are constructed of Oregon pine with hardwood studs and wrought-iron straps and stays. They are lined with 6-lb. lead and have an additional wooden lining of redwood, the bottom of which is in the shape of a tray 6 in. deep. The pine, hardwood and the top board of redwood are dipped in paraffin, which lengthens the life of the tank.

The busbars rest on porcelain insulators, and the dead ends of anodes and cathodes rest on glass blocks. The busbars are of sufficient cross-sectional area to carry at least 500 amp per square inch. The composition of the electrolyte is about 12 per cent $CuSO_4$, 12 per cent free H_2SO_4; the specific gravity is 1.16 to 1.18, and the temperature 120° Fahr. Flow is maintained at the rate of 3 gal. to 4 gal. per minute. A small percentage of soluble chloride is also carried. Current density employed is from 14 amp to 18 amp per square foot. The starting sheets are deposited on rolled copper plates coated with a mixture of oil, black grease and kerosene, or tallow and graphite. In stripping the cathodes care is exercised to withdraw a uniform quantity from the tanks daily, proportionate to the theoretical quantity which should be deposited. The Australian custom is to withdraw from 90 per cent to 95 per cent of the theoretical amount. Anode scrap amounts to 10 per cent to 15 per cent of the anodes loaded. Ninety per cent of the ampere efficiency is obtained.

Anode slime is treated in one of three different ways. First, it may be treated with sodium nitrate and sulphuric acid to dissolve the copper; then with metallic copper to precipitate silver, after which it is washed, dried and smelted with sodium bicarbonate. In the second method the slime is screened and then boiled with sulphuric acid, with subsequent treatment as in the first method. The third method is similar to the first, except that whereas the sodium nitrate and sulphuric acid are previously mixed together before being added to the slime in the first method, they are added separately in the third. In the treatment of the Dore bullion obtained from the smelting of the slime, two of the plants use the old Moebius process of electrolytic refining, while the third uses the Balbach cell.

The cost of electrolytic refining of copper under Australian conditions is given as about 24 s. ($6) for electrolyzing only, per ton of cathode copper, in a plant producing 35 tons per day. To this must be added 2 s. 6 d. ($0.62) per ton for retreatment of scrap. The cost of refining Dore bullion depends very largely on the size of the plant. With a small two-cell Moebius plant capable of depositing 3000 oz. of silver per week, the actual bullion treated was 12,600 oz. at a cost of 0.146 d. (0.3 cents) per ounce. To this must be added the cost of melting, which was 0.103 d. (0.206) cents per ounce, or a total cost of 0.249 d. (0.5 cents) per ounce.

Tin.

Analytical Methods.—Sodium peroxide has proved to be a most useful reagent in analysis, especially in decomposing those minerals which are refractory to ordinary agents. Cleanliness and rapidity are also favorable factors in its use. In the March, 1910 *Journal* of the Chem. Met. & Min. Soc. of South Africa, James Gray gives a résumé of fire and wet methods for assaying tin ores. Fire methods are rightly dismissed as being inaccurate at the best. Of the wet methods (eight being given), the one involving fusion with sodium peroxide, extraction with water and solution with hydrochloric acid, is reported as being the most satisfactory and giving the best results. The details of the method follow. From 0.5 gram to 2 grams of the finely powdered ore are fused from four to six times the same weight of sodium peroxide in an iron crucible. On cooling, the melt is extracted with water. This solution is treated in a flask with an excess of HCl, heated to about 90° C., and reduced with iron nails for 40 to 60 minutes after the solution has become colorless. The flask and contents are then rapidly cooled, a piece of marble added to maintain a neutral atmosphere, and when cold, a little starch paste is added. The solution is then titrated with iodine solution which has been previously standardized against pure tin. A blue color indicates the end-point. The author gives figures showing that the presence of varying quantities of arsenic.

antimony, bismuth, tungsten and titanium have little influence on the accurate results obtained by this method. A bibliography is appended.

Chemical Engineering.

Manufacture of Sulphuric Acid.—The Ducktown Sulphur, Copper & Iron Company, Isabella, Tenn., operates a pyritic smelting plant treating pyrrhotite and manufactures sulphuric acid from the blast-furnace gases. W. H. FREELAND and C. W. RENWICK, respectively, former general manager and present general manager of the company, describe the sulphuric acid plant in the May 28, 1910, issue of the *Eng. & Min. Journ.* Blast-furnace gases from a pyritic smelter are not converted easily into sulphuric acid on account of the varying composition and temperature at which they are delivered, and the flue dust which they contain. The Ducktown company first experimented with the ferric oxide contact process, but found that it was not adapted to the conditions existing in copper smelting. The chamber process, later installed, has proved very satisfactory. The gases to be treated have approximately the following composition: SO_2, 3.5 per cent; CO_2, 3.5 per cent; SO_3, trace. The temperature of the gases frequently varies over 200° C. in 8 or 10 minutes. Flue dust is settled out in flues and dust chambers, and finally in specially constructed dust chambers in which a system of channel irons is arranged to catch the fine particles which have not settled out before. The control of the temperature is accomplished by leading the gases through a greater or less length of a kite-shaped flue leading to the special dust chamber. When the gases are hot they are directed through the entire length of the flue and when cooler through a shorter length. There are four entrances from flue to chamber, through which the gases may pass. Having been thus cleaned, the gases are passed through Glover towers, lead chambers and Gay Lussac towers. A high oxidation of the gas is obtained in the Glover towers, fully 40 per cent being converted into SO_3. The plant has a rated capacity of 160 tons of 60° B. acid per day, but has yielded 200 tons when necessary.

In the *Mining Journal* (London) for April 30 and May 7, 1910, Prof. GIUSEPPE ODDO, of the University of Pavia, discusses the possibilities of manufacturing sulphuric acid from rock sulphur. About two years ago he suggested the idea as a solution of the crisis existing in the Sicilian sulphur market. It was not well received locally because it opposed some traditional customs and empirical ideas. Nevertheless the idea has been worked on and considerable advance has been made in the theory and practice suggested by Professor Oddo. His proposal is to use the fine sulphur rock, which is unsuited for melting, and the poorer grades of melted sulphur which now exist in large stocks on hand, to make briquettes carrying 50 per cent available sulphur. This will utilize one material that is now going to waste and another for which there is little or no demand. The use of sulphur mineral for making sulphuric acid has several notable advantages over pyrites, which Professor Oddo briefly recapitulates as follows:

(a) A purer current of gas which will make possible still further purification by the contact process, and the manufacture of a purer acid for certain purposes.

(b) A current of gas richer in SO_2, in the ratio of about 3:1, which will give proportional economy in plant and operation.

(c) Longer life of the lead chambers in the same proportion and for the same cause.

(d) Facility and regularity of combustion, with an economy of about one-half in manual labor, with 50 per cent briquettes.

(e) Less loss of sulphur in combustion which will only be brought about by a limited formation of calcium sulphide and sulphate. This will be negligible since sulphur will begin to react with the gypsum and limestone in the rock only at a temperature above 420° C., and then very slowly.

(f) Smaller weight of cinder compared with the percentage of sulphur. The best pyrites with 50 per cent sulphur give 68

per cent, while 40 per cent and 45 per cent pyrites give 74.9 per cent and 71.8 per cent, respectively. Briquettes will produce only 50 per cent cinder.

Taking local conditions into consideration, Professor Oddo has worked out the economic features of the industry and has shown that rock sulphur can compete easily with pyrites in the manufacture of acid, and that a concerted action is all that is necessary to open this new industry and put an end to industrial misfortune in the Sicilian sulphur market.

Electrochemistry.

Fixation of Atmospheric Nitrogen.—The old question whether the forced combination of the nitrogen and oxygen in air by means of electric arc discharges is simply a thermal phenomenon (representing the equilibrium at the existing high temperature and independent of the method of producing this temperature) or whether specific electrical effects also play a part in this formation, is again discussed in an interesting paper of Dr. W. HOLWECH in *Zeit. f. Electrochemie*, June 1. Haber and Koenig have formerly shown that at the high temperatures employed in the commercial processes in operation in Norway the reaction seems really to be a purely thermal phenomenon; however, they also have shown that at lower temperatures it is possible to produce more nitrogen oxide than would correspond to the thermal equilibrium. The present author has made experiments with very short direct-current arcs between a cold anode and a hot cathode. He used a water-cooled silver anode and as cathode an iridium sheet which was brought to incandescence by an electric current from an independent source. In order to facilitate the starting of the arc and to reduce the cathode drop of voltage the iridium sheet was covered with calcium oxide. He produced up to 80 grams HNO_3 per kw-hour. The shorter the arc the higher the concentration at minimum speed of the gas. The highest concentrations obtained were 9 per cent NO and more in air. The voltage drop per unit length in the arc was investigated and it was found that although the anode is cold and free from electrode vapor and the cathode is hot, the voltage drop per unit length remains constant within the arc with the single exception of the immediate neighborhood of the two electrodes. It was found that the higher the voltage per unit length the greater is the output and the maximum concentration which can be obtained. This is true as long as the temperature of the arc is not too high. If the temperature is raised the thermic phenomenon hides the electrical phenomenon and it is no longer possible to obtain the same maximum concentration as before. In the commercial furnaces the voltage drop per unit length is less than 10 volts per centimeter. In the author's experiments it was between 100 volts and 200 volts per centimeter. The output in practice is about 60 grams to 75 grams of HNO_3 per kw-hour. The output in the author's experiments is about the same. On the other hand, the maximum concentration which can be obtained is higher in the author's experiments than in practice. The author thinks that in the commercial furnace the thermal phenomenon predominates, while in his experiments the electrical phenomenon predominates in the formation of NO. In his direct-current arcs the temperature was so low that the points of the finest iridium wire did not soften.

Electrolytic Refining of Bismuth.—In the electrolytic refining of lead by the Betts process the anode slime contains silver and bismuth. In *Zeit. f. Electrochemie*, of April 15, F. FOERSTER and E. SCHWABE propose to subject this silver-bismuth to a further refining process in a silico fluoride solution of bismuth as electrolyte. Pure bismuth is deposited on the cathode. The lead contained in the anode passes into the electrolyte. The silver remains back at the anode. The bismuth deposited on the cathode is free from lead.

Alkali Bromide Electrolysis.—A paper by F. FOERSTER and J. YAMASAKI in *Zeit. f. Electrochemie*, April 15, deals with details of the ionic mechanism of the electrolysis of alkali

bromides. Experiments which were made with a well-stirred electrolyte showed that the primary anodic reaction is the discharge of bromine ions, sometimes combined with a discharge of hydroxyl ions. By secondary reactions which are sometimes purely chemical and sometimes electrochemical bromate, etc., is formed. The effects of variations of current density, temperature, concentration of bromide and alkali were investigated, both with platinized platinum anode and with a polished platinum anode.

Potassium Permanganate.—In *Zeit. f. Electrochemie* of March 1 P. ASKENASY and S. KLONOWSKI discuss the electrolytic production of potassium permanganate from solution of potassium manganate. The authors have specially investigated the possibility of getting along without a diaphragm. They used an iron-wire cathode and a large concentric iron sheet as anode. The best amp-hour efficiency was obtained with a cathode current density of 0.9 amp to 0.7 amp per square centimeter in a 7 per cent to 10 per cent manganate solution in about 10 per cent potassium hydroxide. The authors also made some experiments with diaphragms and finally substituted platinum anodes for the iron anodes.

Recommendations are added on the commercial manufacture of potassium manganese. As far as the electrolytic part is concerned it is recommended to use an iron vessel, which serves as anode, and iron cathodes without a diaphragm. The cathodic current density should be 85 amp per square decimeter, the anodic current density 10 times smaller. Distance between anode and cathode about 20 mm; 20,000 amp-hours are required for 100 kg of manganate The voltage is 2.8. About two-thirds of the manganate are changed into permanganate; 1 kg of permanganate requires about 0.7 kw-hour.

Nickel Wire Furnace.—In *Zeit. f. Electrochemie*, May 15, R. LORENZ and G. VON HEVESY record their experience with resistance furnaces using nickel wire. They employ nickel wire covered with asbestos. The life of such a furnace if heated to not more than 700° C. is long, but at higher temperatures the nickel is destroyed. This is due to the formation of nickel oxide on the surface of the wire; this then enters into the interior and the nickel gets brittle and breaks. If higher temperatures up to about 1200° C. are to be used, it is necessary to protect the nickel from the oxygen of the air. This can be done by an air-tight construction.

Recent Metallurgical and Chemical Patents

Gold and Silver.

Cyaniding Apparatus.—The rapid dissolution of gold and silver from complex sulphide and silicious ores, especially those of low grade, is the object of an apparatus patented by P. A. BRANGIER, of Agnew, Cal., and ROYER LUCKENBACH, of Philadelphia, Pa. The general principle involved is the flow of a thin stream of the pulp over inclined surfaces and the application of heat and compressed air at certain points for the purpose of increasing the solvent action of the cyanide. The apparatus is in the form of a box about 10 ft. long and 3 ft. wide and as high as desirable. The box contains a series of superimposed plates inclined in opposite directions from one end of the box to the other. The upper two-thirds portion of each plate is heated by steam and the lower third is convex with air pipes at the sides. As the pulp flows through the apparatus by gravity, traversing first one plate in one direction and the next in the opposite direction, it first flows over the heated portion onto the convex portion. Here the pulp tends to run to the sides of the plate and is thoroughly aerated and scoured by the action of the compressed air issuing from the pipes at the sides. Flowing onto the next plate below, the pulp is treated again in the manner described and so on to the outlet of the box. The succession of plates offers a compact form of apparatus with large operating surface for dissolving the values during the passage of the pulp through the machine. (959,301, May 24, 1910.)

Cyaniding Under Pressure.—Facility and efficiency are claimed for a cyanide barrel recently patented by BENJAMIN O. WEBB, of Los Angeles, Cal. The apparatus consists of an enclosed and revolvable tank or barrel which can be made airtight. It contains a perforated floor which can be used as a filter medium, and has means provided for injecting air or water under pressure. The object of the invention is to provide in one apparatus means for agitating and aerating pulp under pressure, for filtering under pressure when solution of the ore values is complete, and for washing out the filtered pulp. In operation, the ore pulp is introduced through a manhole which is then tightly closed. Through suitable valves oxygen or oxidizing gases are then introduced under a pressure suited to the operation and the apparatus. The barrel is then revolved and the pulp agitated with the compressed oxidizing gases. Projecting knobs on the inside surface of the barrel aid in the complete agitation of the pulp. When this operation is complete the barrel is stopped in what may be called its upright position,

viz., with the manhole up and the inside filter floor horizontal. The operation of filtering the pulp may now be effected by introducing more air and allowing the filtered solution to be drawn off through suitable stop cocks. The pulp may be washed if necessary, after which it is discharged by turning the barrel over and washing the sand out through the manhole by means of streams of water introduced through the same valves used to carry the oxidizing gases. The idea of cyaniding under pressure in the presence of oxidizing gases may add to the efficiency of the cyanide process, particularly with reference to refractory ores. (956,655, May 3, 1910.)

Accelerated Settlement of Slime.—The separation of the solid and liquid components of wet slime or ore pulp is effected by HORACE G. NICHOLS, of Ymir, British Columbia, on the principle of acceleration of settlement due to continuous removal of fine slime as it settles. The accompanying Fig. 1 illustrates the device for this purpose. The slime pulp is introduced through pipe I to a settling chamber B, the walls of which converge downwardly to the opening B directly above an endless traveling belt D moving in the direction shown by the arrows. As the slime settles it is continuously moved upward out of the tank discharged at the uppermost point to which the
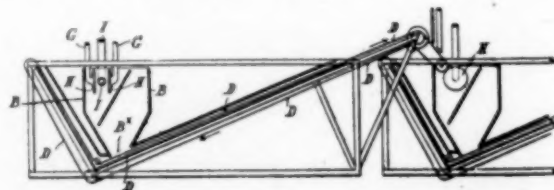

FIG. 1.—APPARATUS FOR THE ACCELERATED SETTLEMENT OF SLIME.

belt travels. The clear, or partly clarified, solution is withdrawn near the surface of the settling chamber B through two suction nozzles H connected with vacuum apparatus by pipes. Provision is made for periodically cleaning the suction nozzles from accumulated slime by means of a reverse current of air or water. The apparatus may be set up in series of two or three, so that the discharged slime from the first may be subjected to a wash and subsequent dewatering in the second and third. This would be the procedure in case slime was being cyanided. The pregnant solution would be withdrawn in the first apparatus discharging slime carrying about 22.5 per cent moisture to the second, where it would be washed and, if neces-

sary, again washed in a third tank. Clear liquid from these weakened cyanide solutions would be drawn off as described above. The apparatus has been in use at the Queen mine in British Columbia, where a plant handles 100 tons of dry slime in 24 hours. (958,272, May 17, 1910.)

Continuous Decantation.—In contrast to the many filtration schemes—vacuum and pressure—for treating sand and slime pulps, the continuous decantation process recently patented by W. L. HOLMS, Guadalajara, Mexico; L. M. GREEN, London, England, and J. A. PATTINSON, Zacatecas, Mexico, is an effort to get away from the necessity of vacuum appliances by making the decantation process mechanically rapid and efficient. The device consists of a series of tanks, say three, with internal mechanism for thickening the pulp. The thickened pulp from the first tank is drawn off at the bottom and pumped to the second and likewise to the third, from which it is discharged to waste. Passing from tank to tank in the opposite direction to the flow of pulp is a supply of wash water which is so introduced as to thoroughly wash the solids free from any dissolved values. The overflow from the third tank is returned as wash water to the second, and from the second to the first. In this way the flow of pulp is against the flow of wash water, the sand or slime being discharged from the bottom of the third tank, while the enriched solution overflows the rim of the first. Each tank contains suitable baffle partitions, agitators, and collectors, and pumps are provided for transferring pulp and solution from one tank to the other. The scheme is ingenious in its detail and may prove to have merit as a means of handling fine pulps. (957,775, May 10, 1910.)

Vacuum Filter.—A recent addition to devices for continuous vacuum filtration is that patented by MARTIN E. HILTNER, of Deadwood, S. D. The invention consists generally of an endless horizontal filter belt divided into sections, traveling around pulleys with means provided for alternately bringing each section into connection with vacuum and pressure pipes. A feed box suspended above the belt at one end of the machine supplies the pulp to be filtered. As each compartment of the belt receives its supply of pulp, a three-way valve connected with the same is actuated by a trip which establishes communication with the vacuum pipe. Each compartment then passes a second trip which actuates the valve so that communication is established with the pressure pipe, through which air or water may be forced to discharge the cake. The filtered solution is piped to suitable tanks prior to recovery of the contained values. (957,558, May 10, 1910.)

Copper.

Treating Impure Matte and Ores.—An impure copper matte, containing lead, is usually a product from most lead blast furnaces. This matte is sometimes partly roasted and then resmelted in the blast furnace, reducing some of the lead and concentrating the copper into a higher grade matte. This process may be repeated until as much of the lead is recovered as is deemed practicable, when the resulting matte is bessemerized. The object of a process recently patented by HERMAN MASCHMEYER, of Hoboken, Belgium, is to eliminate the lead and other impurities and to obtain comparatively pure copper without the repeated roasting and smelting before mentioned. Lead-copper ores may be treated in the same manner as prescribed for the matte. The process comprises two operations: First, a Bessemer roast with added silica; second, smelting in a reverberatory furnace. The first process may be carried out conveniently in Huntington-Heberlein pots. The roast will partially desulphurize the matte or ore and the silica will combine with the lead to form lead silicate. In the reverberatory furnace the lead silicate will go into the slag and remain unreduced. Reaction between copper sulphide of copper oxide will precipitate copper. Thus the separation is designed to be effected in one operation of smelting. Impurities, such as arsenic and antimony, would be partly volatilized and partly slagged off. A further step in the process consists in adding to the mass to be smelted an excess of copper oxide material, which has an oxidizing and refining effect. (957,231, May 10, 1910.)

Tin.

Reducing Cassiterite.—A novel process for the reduction of tin ore, differing radically from the usual process of smelting in reverberatory furnaces with culm, is that patented by ROBERT W. SCOTT, of London, England. The process consists in reducing cassiterite to tin by the action of producer gas. The application of the process is through the use of what may be termed tube mills in which the producer gas and cassiterite are brought in contact with the application of external heat. Reference to Fig. 2 will make the invention clear. An iron tube a is provided with screw caps and tubes at its ends. Stop cocks are provided to govern the flow of gas through the tubes. A furnace is constructed with two compartments, as shown, the first being the fire-box compartment and the second a drying and preheating compartment in which the waste heat from the first is utilized. These furnaces are provided with tubular supports and driving gears into which the iron tubes may be placed. The cassiterite is first dried and preheated in tubes in the
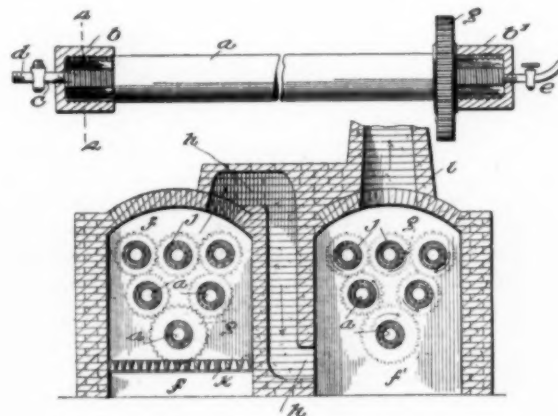

FIG. 2.—APPARATUS FOR REDUCTION OF CASSITERITE.

second compartment of the furnace, after which the tubes are withdrawn from their supports and transferred to the first compartment. Here connection is made between each tube, at the point d, and a source of producer gas. The gas is turned on until it will burn at the point e. The tubes are then set in rotation and kept revolving for the necessary length of time to complete the action of the gas on the cassiterite. The gas is kept under a sufficient heat pressure to keep the tubes constantly full of gas. Metallic tin is the product of the reaction. (957,111, May 3, 1910.)

Zinc.

Regenerative Furnace.—An improvement in furnaces for zinc distillation of the type generally known as "reverse regenerative," or more properly "reverse recuperative" type, is the subject of a patent granted to Dr. FRANZ MEYER, of Englewood, N. J. (now of Uerdingen, Germany). The principal feature of the invention consists in dividing the combustion chamber into two long flue-like compartments which communicate with each other at one end of the furnace. This permits of a complete reversal of the flow of air for combustion and of the products of combustion through the two compartments of the furnace proper, and through the recuperative flues. Fig. 3 represents a vertical cross-section of the furnace and Fig. 4 a horizontal longitudinal section taken just below the retorts, or on the floor of the combustion chambers. In Fig. 3 AA' and BB' are the recuperative flues through which air for combustion may be directed alternately by means of a reversing valve at the inlet, not shown; ee' are flues conducting the air from the recuperative flues to the combustion chambers

CC' (Fig. 4). In one position of the reversing valve, which may be of the Forter type, air enters through the recuperative chambers AA', and by way of e into the combustion chamber C, where it mingles with gas issuing from the jets n in the first combustion chamber. It then passes through the communi-cating passage p into the second combustion chamber C', where additional combustion of gas takes place. The products of combustion then pass through e' and through the recuperative chambers BB'. When this course has been pursued long enough to abstract the heat from the recuperative chambers AA' and to heat the chambers BB' by the waste gases, the flow of air is reversed. In this way the air for combustion is preheated alternately in the recuperative chambers. The gas for combustion is preheated by passage through the chambers ll' l'. (958,150, May 17, 1910.)

FIG. 3.—VERTICAL SECTION OF REGENERATIVE FURNACE.

Nickel.

Reduction of Nickel Hydrosilicate Ores.—Mr. ADOLPHE CHALAS, of Philadelphia, Pa., patents a process of reducing sili-cious ores, such as garnierite, which contains nickel as a hydro-silicate together with a large proportion of magnesia. The other reducible metals contained in such ores are iron and a very small proportion of cobalt, manganese, and chromium, while the other refractory gangue materials, besides silica and magnesia, are very small proportions of alumina and lime. The hydro-silicate ores are characterized by their high combined water contents, averaging about 10 per cent by weight. They very easily absorb water, being of a clayey texture, and always retain a large amount of hydroscopic water, averaging over 20 per cent by weight. The present process essentially comprises three steps.
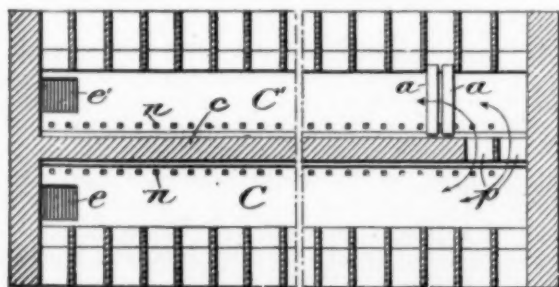
FIG. 4.—HORIZONTAL SECTION OF REGENERATIVE FURNACE.

The first preparatory step consists in briquetting a mixture of the nickel ores, serving as a binding material, with fine ores of high iron contents, such as magnetite sands and concentrates, flue dust, iron scale, purple ore, or the leached residues of roasted nickel copper pyrites.

In the second step metallurgical treatment begins. These bri-quettes are reduced in an ordinary blast furnace, with a suffi-cient quantity of lime to give a basic slag, producing a nickel-iferous pig iron which is largely separated from the slag by gravitation.

Finally, the third stage of the process consists in chilling the molten slag by running it into water, crushing it, and passing it with water through a magnetic separator, by which practically every particle of nickel-iron alloy is recovered. The residual slag contains no nickel, owing to the lack of affinity of this metal for anhydrous silica, and to the fact that the iron, being more oxidizable than nickel, protects the latter from oxidation and scorification. (959,460, May 31, 1910.)

Furnaces.

Ore Roasting Furnace.—The reverberatory type of roast-ing furnace, both hand and mechanically rabbled, has been the subject of many patented improvements of various value. Let-ters patent have recently been granted to C. E. MARK, of Chica-go, Ill., for an improved hearth claimed by the inventor to promote the roasting process and render it more efficient. As described by the inventor, the hearth is made of fire brick, or other suitable refractory material, perforated at regular inter-vals to permit a slight blast of air to be admitted from below while the charge is being rabbled either by hand or mechanical means. The idea is that by this intimate contact of air with the roasting ore, the oxidation of the latter will be more rapid and complete than in the usual form of furnace. The inventor's furnace may be oil-fired if desired, or gas may be used. Prac-tical operation would determine whether such a furnace would make an excessive amount of flue dust, and whether the aper-tures in the hearth could easily be kept free from agglomera-tions of sintered ore. (952,680, March 22, 1910.)

Melting Furnace.—In the type of furnace known as the "pot furnace," used for melting metals and alloys, ALFRED FISHER, of Chicago, Ill., has made certain improvements. In one instance the improvement consists in providing means for the even distribution of heat to the pot by means of a flame directed in an ascending spiral direction. Provision is made for an intimate mixture of air and gas, which passes around the furnace several times. The object of a second invention is to provide in a practical manner for the operation of a plurality of melting furnaces, so that the heat can be regulated in each and all of the furnaces independently of the operation or non-operation of any one of them. Provision is also made for the ready removal from the bottom of the furnace of a broken crucible or molten metal resulting from the breaking of a crucible. (957,873 and 958,384, May 17, 1910.)

Dross Furnace.—For the purpose of recovering metal from dross, GEORGE L. WALKER, of Pueblo, Col., has devised a furnace in which the dross receptacle consists of a conical caldron designed to be rotated on its ends as trunnions. The caldron is contained in any suitable form of coal or gas-fired furnace, and is thus kept within the heated interior without being exposed to the open air. A conical stopper holds the dross in place in the caldron, which is revolved so as to facilitate the settlement of globules of metal from the dross. As the metal gathers it runs out of the lower end of the caldron. In this form of dross furnace, oxidation by the air is reduced to a minimum and the sweating effect is magnified. (949,311, Feb. 15, 1910.)

Gas-Cleaning Apparatus.

Smoke and Gas Condenser.—An improved form of con-denser for use in connection with smoke, gases and vapors, which the inventor states has given satisfaction in practice, is the basis of a patent of SAMUEL G. MERRICK, of Philadelphia, Pa. The apparatus consists of a casing of metal, or other suitable material, divided by a partition into a receiving and a delivery chamber having respectively an inlet and outlet for gas. A shaft is mounted for rotation in this casing, and carries a tubular member having a spiral partition running through it. In Fig. 5 the description thus far given will be apparent. The receiving end of the revolving tubular part has the shape of a frustum of a cone and is adapted to receive a large volume of gas. As the tube rotates and the gas is advanced toward the delivery end, it is compressed into smaller volume by reason of the constricted size of the tube at that end. This is an important feature of the invention in that it continues the com-pressing and purifying which would otherwise cease at the end of the frustum. Baffle plate 13 is arranged as shown for the purpose of breaking up the liquid in the tank and effecting a thorough mixture with the gas. In operation the casing is partly filled with water as shown. Smoke, gas and vapor enter

through the pipe 3 and are conveyed through the revolving tubular member. During its confined passage the gas is subjected to a thorough washing and condensation and issues free
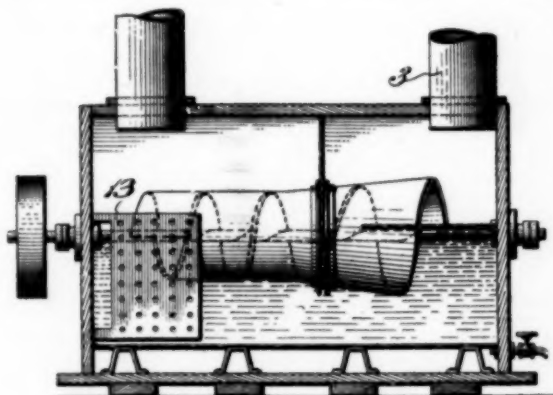

FIG. 5.—SMOKE AND GAS CONDENSER.

from solid particles and soluble constituents. (946,146, Jan. 11, 1910.)

Cleaning Blast-Furnace Gas.—Gravity and centrifugal action are the forces utilized in a device for cleaning gases from iron blast furnaces, according to a patent granted to JOSEPH MORGAN, of Westmore Borough, Pa. The device might also be applied to the removal of valuable flue dust from the gases from other metallurgical furnaces. Provision is also made for reheating the gas before use, and for washing it if necessary. The details of the cleaner may be seen in Fig. 6. The gas is first drawn from the top of a furnace down to a settling chamber, where, by reason of the expansion and decreased velocity of the gas, the larger dust particles separate out by gravity and may be withdrawn at the bottom of the chamber. From here the gas is led to the cleaning apparatus illustrated. Entering at 8 the gas passes through a spiral passageway formed by a helical partition. During this passage the particles of dust will fall into traps 15, where they will remain
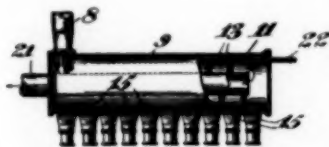
FIG. 6.—GAS CLEANER.

until withdrawn through suitable gates or valves. During this passage, also, the gas may be sprayed with water from the pipe 22, both water and solid particles then settling into the traps 15. Having passed the length of the spiral passageway, the gas now returns through a central pipe as shown by the arrow. Here the gas is reheated prior to use. (948,062, Feb. 1, 1910.)

Miscellaneous Metallurgical Processes.

Basic Lead Sulphate Pigment.—The manufacture of basic lead sulphate pigment by volatilizing and oxidizing galena ores has usually resulted in a yield of about 80 per cent of a product which has not always had a good color. In addition to these drawbacks a certain amount of lead is usually reduced, thus making the process inefficient in the points of yield and color of product. J. B. HANNAY, of Cove Castle, Scotland, and S. B. WILSON, of Westminster, London, England, have been granted patent on a process for converting lead sulphide ores into basic lead sulphate, in which they claim a high yield of good white pigment of constant composition, without attendant reduction of pig lead. In a blast furnace supplied with a hot-air blast a column of coke is heated to about 1550° C. with a temperature at the top of the furnace of about 1250° C. Galena is fed onto this hot mass, with the result that globules of galena become surrounded with an atmosphere of the same composition. The furnace being thus filled with vapors of lead sulphide and carbonic oxide, and the liquid galena being surrounded by gaseous lead sulphide, there is no opportunity for reduction of lead.

The mixed vapors pass from the furnace to a combustion chamber where a large excess of cold air is admitted. Combustion of the lead sulphide and carbonic oxide ensues and cooling is effected by the large excess of air. The lead fume now passes to a water-cooled condenser where it is collected in conical hoppers. The inventors have provided means for feeding the furnace with hot dry coke and dry galena in such a manner that the balance of flame temperature in the apparatus is not disturbed by feeding. They state further that they have obtained 111 parts of basic sulphate of a composition between $3PbSO_4 \cdot PbO$ and $2PbSO_4 \cdot PbO$ from 100 parts of galena containing about 5 per cent of impurities. The product is ready for use and requires no subsequent treatment of any kind. A furnace 36 in. internal diameter will produce 4 tons of lead sulphate in 24 hours. (955,396, April 19, 1910.)

Briquetting Fine Ore.—Flue dust, fine ore, waste iron, etc., are prepared for smelting by teatment in one of the methods recently patented by MAX GLASS, of Vienna, Austria. In the first process, the fine material is mixed with a solution of calcium chloride and water glass, when the following chemical reaction occurs:

$$Na_2SiO_3 + CaCl_2 = CaSiO_3 + 2NaCl$$

The silicate acts as a binding agent. Borax may be added to briquettes of fine iron metal to prevent rusting. The reagents are used in the following strengths and proportions: 55g $CaCl_2$ to one liter water, 175g Na_2SiO_3 to one liter water. The borax solution will contain 40g to 50g per liter. The second process of the same inventor contemplates the use of calcium acetate and aluminium sulphate, with an admixture of quicklime and borax if desired. The reactions are as follows:

$$3Ca(CH_3COO)_2 + Al_2(SO_4)_3 = 3CaSO_4 + Al_2(CH_3COO)_6$$
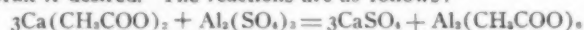
If lime is present, considerable heat is formed by its combination with the water, and under the action of this heat aluminium acetate is further converted into an insoluble basic salt which has a binding action. The reagents used in this process are: 28g CaO mixed with 60g CH_3COOH to slightly acid reaction, and diluted to one liter with water. 60g $Al_2(SO_4)_3$ are dissolved in one liter water, to which are added 50g to 60g powdered lime. Borax solution may be added as indicated above. One liter of the binding agent is mixed with every 100 kg of the material to be briquetted. The process produces briquettes which need no further heating. (958,700-1, May 17, 1910.)
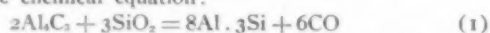
Electric Furnaces.

Purification of Bauxite.—Mr. Lewis E. Saunders, of the Niagara Falls works of the Norton Company, has devised a process in which he makes use of the strong reducing properties of aluminium carbide to secure a nearly complete elimination of the metals or impurities associated with the alumina, and he also makes use of the known fact that silicon is capable of preventing the absorption of large proportions of carbon by such metals as iron and titanium. In his process of purifying silicious bauxite or clay he adds to it a proportion of carbon, largely in excess of that required to reduce the iron, silicon, and titanium therein, and fuses the mass in an electric furnace, whereby a considerable portion or even all of the alumina is converted into aluminium carbide. This carbide is capable of mixing with molten alumina or of dissolving or diffusing therein, and when so dissolved or diffused, or when in a molten condition, is a most energetic reducing agent, by the action of which, supplementing that of the carbon, the oxides of the associated impurities are reduced with substantial completeness, at the same time the reduced metal or alloy is by its content of silicon prevented from absorbing or combining with such proportion of carbon as would bring about an amount of resolution of diffusion capable of contaminating the bath to a considerable degree. The reduced metals are allowed to settle from the molten bath of aluminium carbide, or alumina and aluminium carbide, and are removed therefrom by tapping. The molten bath, substantially free from metals other than aluminium, is then oxidized, for instance, by blowing air through

the molten mass in the furnace or after tapping by showering the molten mass through air. This reaction is strongly exothermic and the fluidity of the bath is increased by the heat set free, whereby the operation of blowing and any pouring or tapping operation is rendered easier. The result is that a homogeneous mass of commercially pure alumina is obtained. (960,712, June 7, 1910.)

Reduction of Aluminium Ore.—A process for the reduction of aluminium ore is described by Mr. Frank J. Tone, of the Carborundum Company, of Niagara Falls. It consists of two steps. In the first step the aluminium ore is mixed with the proper amount of carbon and reduced to aluminium carbide. In the second step the aluminium carbide is used as a reducing agent in place of carbon for the reduction of ores of various elements to alloys of aluminium and these elements or for the reduction of aluminium ores to elemental aluminium as the case may be. For example, in the manufacture of a silicon-aluminium alloy corresponding to the formula $8Al.3Si$, aluminium carbide and silica are taken in the proportions expressed by the chemical equation:

$$2Al_4C_3 + 3SiO_2 = 8Al.3Si + 6CO \qquad (1)$$

and subjected to heat in an electric furnace, obtaining the silicon-aluminium alloy as the final product. If the element aluminium is to be produced as the final product a mixture of aluminium carbide and aluminium oxide is made in accordance with the following equation:

$$Al_4C_3 + Al_2O_3 = 6Al + 3CO. \qquad (2)$$
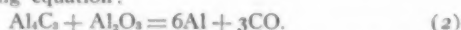
The advantage of dividing the process into two parts is that the reduction of aluminium ores to aluminium carbide can be accomplished with much greater efficiency than the direct reduction of the ore to the element aluminium. The reason for this is that furnace processes can be adopted for the production of the carbide which are highly efficient and entail little or no loss by volatilization. Of the six parts of aluminium produced in equation (2), four parts are the result of the highly efficient first reduction to the carbide and two parts are the result of the less efficient second step of the process. The amount of CO gas evolved per unit of aluminium produced in equation (2) is also much less than in the reaction in which the oxide is directly reduced by carbon. (961,913, June 21, 1910.)

Graphitizing Carbon.—A process of Mr. Frank J. Tone of the Carborundum Company, relates to the "electric heating of carbon, for the purpose of baking, purifying, polymerizing, graphitizing, or otherwise changing its character by heat treatment." The essential feature is the use of an arc furnace instead of a resistance furnace. In Fig. 1 the charge of carbon to be converted into graphite is indicated by 3, while 4 and 5 are the two electrodes. The upper electrode is surrounded by a shield 6 of carbon, which is insulated from the electrode by a refractory insulating layer 8. This shield 6 prevents contact between the charge and the electrode so as to avoid shunting of the current around the arc. An increase of efficiency over resistance heating is claimed, due to the greater amount of energy which can be generated and supplied in an arc furnace to a given volume of charge in a given time. Not only is the carbon converted in graphite, but any impurities, such as silica, alumina, iron, calcium, etc., are volatilized. (961,912, June 21, 1910.)

Electrode Connection.—Mr. Charles A. Keller, of Paris, France, has devised the following method of making connection between a carbon electrode and the conductor. The metallic conductor comprises a cast-iron or steel sheath enclosing a hollow copper core through which cooling water circulates. A recess is formed in the head of the carbon electrode and in this recess the end of the conductor is inserted, the conductor being smaller than the recess in the electrode. Molten copper in then poured into the recess around the end of the conductor, which is thus physically connected in a permanent manner to the electrode. Recesses are formed in that portion of the conductor which lies within the carbon electrode and

these recesses are entered and partly filled by the molten copper, "an improved contact being thus obtained and air gaps being provided which tend to prevent the formation of induced currents." (961,139, June 14, 1910.)

Alloy for Resistor Wires.—Mr. Wilbur B. Driver, of East Orange, N. J., patents the addition of aluminium (1 to 9 per cent) to alloys of at least two other metals for the purpose of producing wire for resistor purposes; for instance, to copper-nickel alloys, copper-manganese alloys, copper-nickel-manga-
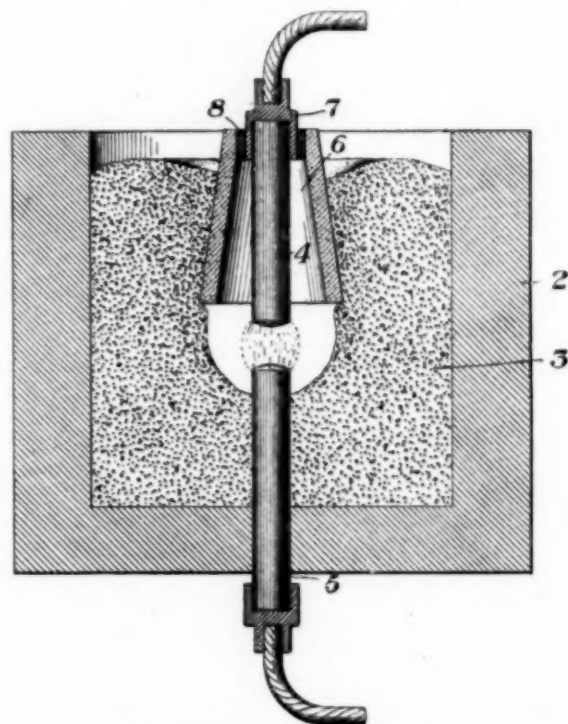

FIG. 7.—ARC FURNACE FOR GRAPHITE.

nese alloys, and iron-nickel alloys. An alloy containing copper and manganese and 5 per cent of aluminium and an alloy containing copper, nickel, and manganese and 5 per cent of aluminium, are especially mentioned in the claims. (961,217, June 14, 1910.)

Electrolytic Processes.

Metallic Sodium.—Mr. Robert J. McKnitt, of Niagara Falls, proposes the following method of producing sodium from sodium chloride (assigned to Roessler & Hasslacher Chemical Company). The sodium in sodium chloride is first alloyed with lead or tin, by some such method as electrolyzing fused sodium chloride with a fused lead cathode, as in the Acker process. The lead-sodium alloy is then heated in a partial-vacuum chamber under reduced pressure. The sodium distills off and is condensed in a separate condenser. The process may be made continuous by conveying the molten alloy from the electrolytic furnace to the receiver, distilling off the sodium and returning the remaining lead to the furnace. In this case but little heat will have to be supplied to the receiver, as the temperature of the alloy in the furnace is usually about 800° C., while sodium in vacuo boils off readily below 800° C., it being necessary simply to maintain the temperature while reducing the pressure. Of the claims which are broad and do not refer to sodium, the first one may be quoted. It relates to "the method of reducing a metal from its compounds which consists in dissolving, alloying or chemically combining said metal with another metal or mixture of metals less volatile than the metal desired, submitting the resultant solution, alloy or compound to a pressure lower than that due to the atmosphere and to a temperature suitable for the distillation of the metal desired." (959,785, May 31, 1910.)

Cathode for Diaphragm Cells.—A patent of Mr. ELMER A. SPERRY (assigned to Hooker Electrochemical Company) relates to a metallic cathode plate for use with such electrolytic cells in which a diaphragm is placed in contact with one face of the cathode, as in the Townsend cell. In such cells, and particularly in case a high current density is employed at the cathode, it is desirable to secure a substantially even distribution of the current over the face of the cathode plate, as otherwise irregularities of action may ensue, resulting in a decreased efficiency of operation. The perforations of the cathode should preferably aggregate 50 per cent or more of its exposed area. These perforations have heretofore usually comprised uniformly distributed circular apertures. While the electrical conductivity of a plate of this character is substantially uniform in all directions, it has been found that such apertures tend to clog or fill in practice when in contact with impure liquids, the deposit accumulating in the lower portion of each aperture to a height dependent upon the thickness of the plate; after a certain amount of material has collected, any further deposit is washed out and eliminated by the escaping gases and percolating liquid. The difficulty may be largely overcome and other important advantages secured by elongating the apertures in a vertical direction. In this case the deposit, being dependent upon the thickness of the plate, does not rise higher than in a circular orifice, and thus clogs or fills a much less proportion of the area of the perforations. These elongated perforations or slots possess a further advantage which is independent of their vertical or other position, in that for a given percentage area of aperture in the plate they afford, as compared with circular apertures, an increased active cathode area; this results from the fact that the walls of the perforations are cathodically active to the extent that they are wet with the brine or other solution undergoing electrolysis, or are in contact with the saturated diaphragm; and a far greater length of wall is provided by the use of the elongated apertures. It is desirable that the cathode plate should be as thin as is consistent with strength and the considerations of electrical conductivity. In order that it may be made thin and at the same time may effect a substantially even distribution of the current it is essential that the current should be conducted from the cathode at those edges which are transverse to the slots or elongated perforations. In this way a direct and short path is afforded for the current from any portion of the plate to the conductors. (961,549, June 14, 1910.)

Gold Refining.—An important modification of the well-known Wohlwill gold refining process (in use in the United States mints and elsewhere) is patented by Dr. HEINRICH WOHLWILL, of Hamburg, Germany. The essential feature is that the development of chlorine at the surface of the anode can be entirely prevented by employing a non-symmetrical alternating current (a superposition of direct and alternating current) instead of the usual pure direct current. The process is therefore made applicable to the refining of gold carrying a large percentage of silver. Other advantages are a materially shortening of time and reduction of expense. The process was already described in some detail in our February issue, page 82. (961,294, June 21, 1910.)

Miscellaneous.

Tungsten Powder.—Metallic tungsten in powder form is obtained by reduction of tungstic oxide by the following method of Dr. ELIHU THOMSON (assigned to General Electric Company). The tungstic oxide powder is fed uniformly from a hopper into an inclined heated tube in which it is moved forward and downward by a revolving Archimedean screw. Air is kept out of this tube and hydrogen gas is passed through it in a direction opposite to that of the movement of the powder. At a temperature between 600° C. and 800° C. reduction takes place according to the equation $WO_3 + 6H = W + 3H_2O$. Only a small part of the hydrogen in the tube is used up in this reaction; the hydrogen-steam mixture having been withdrawn is

therefore freed from the water vapor first by condensation in a water-cooled condenser and then by contact with metallic sodium, hydrogen being regenerated in the latter reaction from the steam. The unused hydrogen, together with the regenerated hydrogen, is carried back into the reduction tube. (960,441, June 7, 1910.)

Treatment of Wool.—Messrs. R. VANDATTE and B. LAGYE, of Antwerp, Belgium, patent details of an apparatus for removing the yolk and the grease from wool, the wool being subjected to the action of an electric current while it is moved about in a vat filled with yolk water. This process differs from similar ones by the electric current being conducted to the wool under treatment through rakes which, while their supports convey the current to the wool, also insure the carrying of the wool along the vat and its subdivision or opening out therein. (961,468, 1910.)

Ageing Liquors.—Mechanical details of an apparatus in which a liquor is simultaneously subjected to the action of gases and to the action of an electric current for the purpose of ageing, are the subject of a patent of Mr. J. SEITZ, of Chicago. (961,167, June 14, 1910.)

Recent Progress in Wood Distillation.

By M. KLAR.

In the March issue of this journal (Vol. VIII, p. 155) Mr. Charles L. Campbell discusses the wood-distilling industry in the United States, with reference to present methods, market prices of the products, prospects of the manufacture of acetic acid and methyl alcohol from the crude products of wood distillation, and finally gives a review of improvements which have been devised and introduced by me and my firm, F. H. Meyer, Hannover-Hainholz, Germany, during the last years in wood distillation in various countries, including the United States and Canada.

The information given there was essentially based on my book, Technologie der Holzverkohlung (2d edition, 1910; published by Julius Springer, Berlin) and on prospectus 290 a to 290 e, 1909, of the firm of F. H. Meyer. But there are some inaccuracies on practical details, which I wish to correct.

Thus my tar separator (American patent 868,347) has not yet been used in practice for kiln plants, because the enormous quantities of gas in this process would necessitate the design of apparatus of enormous dimensions and therefore very high first cost.

To treat the crude wood vinegar of the kiln plant for the production of acetate of lime and wood alcohol, I employ a continuous distillation process, which differs, however, from those used heretofore in the United States in so far as the latent heat contained in the first distillation is repeatedly utilized. The result is that the cost of distillation with the methods usually employed at present, is reduced by about 70 per cent.

The statement of Mr. Campbell that my continuous drying apparatus for gray acetate of lime is operated with superheated steam, is also incorrect. In my evaporation and drying process (which is the first entirely continuous and mechanical process in practical use) nothing but exhaust steam is employed, so that the production of the gray acetate of lime from its solution costs practically nothing.

In the following I intend to supplement Mr. Campbell's article by showing in detail the relative advantages and disadvantages of the processes hitherto used in the United States, and my new processes.

Distilling of Wood.

Three methods have hitherto been practised in the United States, employing kilns, retorts or ovens respectively.

(a) Kilns have the disadvantage of insufficient yield.

(b) With retorts, that is, small horizontal iron retorts, the yields are poor, the fuel consumption high, and wages high.

(c) In ovens, that is, in large horizontal iron retorts,

charged with wood with the aid of carriages, good yields are produced, but the costs of repairs are very high, on account of the unfavorable profile of retort and carriage, while the fuel consumption is also high.

On the other hand, in the new process of F. H. Meyer, large horizontal retorts of 8 to 9 cords capacity are only used, charged by carriages. However, the retorts and the carriage have a profile which offers a far larger resistance. For heating wood, producer gas is used, whereby not only the whole length of the retorts is uniformly heated, but a certain amount of acetate of lime and wood naphtha is obtained as by-products from the fuel (wood). These by-products cover the cost of the fuel. Coal producer gas, furnace gas, natural gas can be used instead of wood producer gas. The preheating of the gas as well as of the air used for burning, and of the iron carriages with the charge of wood, entails no expense with the F. H. Meyer retort.

This system has the great advantage that repairs of carriages and retorts are very rare, even after many years of working. By the regular heating with gas, a considerable saving of fuel, an easy regulation of the fire, and thorough regularity in the heating of the retort are obtained. The length of life of the retort is thereby prolonged; besides the yields are increased. Through the production of byproducts out of the fuel, in case wood producer gas is used, the fuel really costs nothing, since these byproducts cover the expenses for the fuel. Besides the saving of fuel, the retorts are preserved, the wood charring process is shortened, and the production is increased.

Distilling of Crude Liquor.

In the process hitherto used in the United States, the cooled crude liquor was distilled in simple copper stills, connected with a condenser. This distillation has in view the removal of the tar, dissolved in the crude liquor, in order to produce acetate of lime of 80 per cent.

The disadvantage of this usual method is that considerable quantities of steam are required for the distillation, amounting almost to 50 per cent. of the whole quantity of steam, which is required in wood distillation.

On the other hand the process of F. H. Meyer employs two methods for separating the tar from the crude liquor.

(a) Between the retort and the condenser the tar separator (American Patent No. 868,347) is inserted, whereby the whole quantity of tar, which is contained in the wood distillation products, is removed before the condensation of the crude liquor. This one can be neutralized on leaving the retorts without a special distillation.

(b) The crude liquor is distilled by connecting several distilling apparatus with each other, and these are kept under different pressures. The latent heat of the vapors formed in the first apparatus distills the crude liquor in the second apparatus and the vapors of the second apparatus distill the crude liquor of the third apparatus.

The advantages of these methods may be summed up as follows:

(a) By the apparatus (a) 868,347, which is already in operation in the United States, the whole consumption of steam which heretofore was required for the crude liquor distilling, is rendered unnecessary. The disagreeable cleaning of the primary stills and the retort condensers is also done away with. Further, there is a considerable saving of fuel, wages, apparatus and space.

(b) The apparatus (b) can be worked by waste steam, and in this case do not require any fuel at all, if sufficient waste steam is to be had. In case no waste steam is available, the apparatus permits a saving of fuel of 63 per cent., compared with the consumption in former American practice.

Manufacture of Crude Wood Alcohol from Neutralized and Distilled Crude Liquor.

In the process used so far in the United States, this was done by distilling the neutralized liquor in stills until the distillate became free from alcohol. In order to accomplish this, 30 per cent. had to be distilled off, which contained about 10 per cent. of wood alcohol. After a second or a third rectification of this weak wood alcohol, the commercial product was obtained.

The disadvantage of this method is that the three successive periodical operations require a heating to the boiling point each time; they require a large number of apparatus, much supervision and attendance, and involve a very great consumption of steam, while the losses are considerable.

On the other hand, in the new process of Mr. F. H. Meyer the solution of acetate of lime containing wood alcohol, which is obtained by neutralizing the distilled crude liquor, is separated in a single continuous operation and with only one apparatus into a commercial crude wood alcohol and a solution of acetate of lime free from wood alcohol, while without expense the acetate solution containing wood alcohol is preheated before entering the apparatus.

The advantages of this new method are considerable saving of steam, considerable saving of space, considerable saving of wages and simplification of the working, and practically no losses.

Evaporation of Acetate of Lime Solution.

This was hitherto done in open pans with double bottom, heated until a crystalline paste was formed. In this process crusts form on the heating surfaces.

The disadvantage of this method is that the forming of crusts results in an exceedingly high consumption of steam which is not at all in proportion with the evaporated quantity of water. Further, considerable manual labor is necessary in taking out the crystallized paste and removing the crusts. For this reason these open evaporating pans are a very expensive and very troublesome apparatus in a wood distillation plant.

On the other hand, in the new process of Mr. F. H. Meyer the evaporation of the acetate of lime solution does not take place in open pans, but continuously in closed vessels, and only to such an extent as not to allow any crystals to separate. Thereby the vapors of one evaporator heat the solution in the other one, so that in connecting three evaporators, one needs to supply steam only to the first one. The further evaporation of the concentrated solution of acetate of lime, with formation of crystals, is carried out in revolving apparatus; the acetate is dried on its surface and is continuously removed by mechanical scrapers and transferred continuously to the final dryer.

The advantage of this new process is that in using several evaporators which work in combination together, one needs only about 37 per cent of the steam hitherto required for the concentration, as the latent heat is utilized three times. The apparatus works continuously, needs but little space, and not much of attendance. The heating surfaces of the revolving apparatus are constantly kept free from crusts by the scrapers. The saving of steam and wages is considerable, and the troubles of taking out the crystals and scraping off the crusts by hand are entirely done away with.

Drying of Acetate of Lime.

This was carried out heretofore on an open drying floor, heated by direct fire or fire gases with manual labor. The chief disadvantages are that a great many workmen were required, who usually do not like the work, so that there is the constant trouble of finding new labor. As a matter of fact, the unpleasant smell is very annoying and the workrooms are covered with dust.

On the other hand, in the new process of Mr. F. H. Meyer this final drying operation is carried out in a continuously working apparatus, the dried acetate of lime being directly put into sacks without any intervention. The evaporator, the revolving dryer, and the final dryer work together in an abso-

lutely continuous matter; the acetate of lime solution enters the evaporator and the dried acetate leaves the final dryer, there being no manual labor.

This continuously working apparatus necessitates even for the largest plants only one man per apparatus. The disagreeable work on the open dryer is avoided; the work is a pleasant one and the required space is greatly reduced. The acetate of lime enters the apparatus on one side and leaves the same on the other side as acetate ready for the market, without any manual labor and using waste steam only for heating.

The following table gives a comparison of the consumption of fuel and the number of workmen needed in the older process used in the United States and in the new process of Mr. F. H. Meyer.

The table shows that a wood distilling plant operated by the system of Mr. F. H. Meyer is altogether different from

bonization. This should prove a great advantage in many places.

To sum up the characteristic features of the new Meyer-Klar wood distillation process, the following types of apparatus are used and have proven successful in commercial practice:

(1) Retorts of 8 cords capacity, which are charged and emptied by means of carriages and are heated by gas. The method of firing has been carefully worked out in practice, wood-producer gas or any other gas being employed. The whole length of the retort is heated perfectly uniformly. On account of the special profile of the retort the troublesome repairs, resulting in interruption of operation, are reduced to a minimum. The cost of operation is reduced, the life and the yield are increased, and the fuel consumption is very materially decreased.

(2) Use of preheaters for the wood which is to be sub-

NATURE OF PROCESS	Comparative Fuel Consumption		Comparative Labor Required	
	Old Process	Meyer Process	Old Process	Meyer Process
1. Distillation of wood in large retorts (ovens).				
2. Distillation of crude liquor.				
3. Manufacture of crude wood alcohol from acetate of lime solution.				
4. Evaporation and drying of acetate of lime solution.				
Summary giving the comparative totals.				

former plants. This is especially true if wood-gas producers are employed; they originated in Hungary, but have been considerably improved. These apparatus produce the heating gases for the retorts and the boiler out of the cheapest waste wood. The producers are continuously charged with wood. While the wood falls down, the gas is gradually removed and the whole carbon, which the wood contained, is changed into producer gas, whereby the wood is carbonized at the same time by the hot generator gases.

The products escaping from the gas producer consist of crude liquor vapors, wood gas, and wood producer gas. These products pass through a condenser in which the crude liquor is condensed, which is worked up to acetate of lime and wood alcohol. The remaining gases are utilized, as mentioned above, after being preheated without cost, to heat the retorts and the boiler. We have, therefore, simply an apparatus heated by gas with all its advantages. The quantity of gas obtained by this operation is large enough to work up the crude liquor out of the producer itself as well as the crude liquor coming out of the ovens or retorts.

With these apparatus the fuel costs practically nothing, if the price of the wood is not too high. They are specially useful in using cheap kinds of wood, which are not fit for car-

jected to distillation. In these apparatus the wood is preheated and dried without expense. Other advantages of this simple and cheap method are a smaller consumption of fuel in the distillation which follows, and increase of output of acetate of lime, wood alcohol, and charcoal.

(3) The use of special devices in the chambers for cooling the charcoal, whereby the time of cooling is reduced to 24 hours, while only one cooling chamber is needed instead of two or three as heretofore.

(4) The use of preheaters, the operation of which entails no expense, for preheating the gases and air before entering into the fireplace, whereby considerable quantities of fuel are saved.

(5) The use of wood-gas producers, operated with waste wood. They furnish not only the gases required for heating the retorts and boilers, but it is also possible to recover the acetate of lime and alcohol in sufficient quantities to pay for the wood consumed in the producer. Hence, there is practically no expense for the heating—that is, in this case for the fuel for the whole installation.

(6) Washing of the wood gases and the heating gases before combustion, whereby the output is increased.

(7) Separation of the tar before the condensation of the

wood vinegar vapors by means of Klar's tar separator. This renders unnecessary the usual distillation of the wood vinegar in primary stills, so that the main quantity of the heating vapors used in a distillation plant is saved.

(8) Use of wood vinegar distillation apparatus, repeatedly utilizing the latent heat, instead of primary stills, in cases in which tar separators cannot be employed, as, for instance, in kiln plants. These apparatus operate absolutely continuously and, compared with primary stills, they permit a saving of 70 per cent of steam, if steam under pressure is used, or of 100 per cent if exhaust steam is used.

(9) Use of continuously operated rectification apparatus for the production of crude wood naphtha from the calcium acetate solution. In a single operation (instead of three or four, as employed heretofore) commercial crude wood naphtha is produced, of about 90 per cent, mixible with water.

(10) Use of apparatus operated perfectly continuously and mechanically for producing dry acetate of lime from its solution, with the use of only steam or exhaust steam for heating. The solution of this problem has been tried repeatedly in this country, but always without the desired success in spite of considerable expense of money. After many years of experimental work I and the firm of F. H. Meyer have solved this problem, and these purely mechanical lime acetate dryers have now been in successful operation for a number of years in quite a number of works. Those who know the difficulties and complications of present methods of drying lime acetate solutions and the high cost of fuel and labor will appreciate the advantages of our apparatus, which is supplied at one end with dilute acetate solution and which yields at the other end commercial gray acetate of lime directly into the bags, while only a relatively small percentage of the heat employed in usual processes is needed in the new process, and one or two men in a shift can take care of the drying of more than 30,000 lb. per day.

All the processes and special apparatus noticed above have been developed by us in almost 15 years of work in wood distillation. All of them are in practical operation in numerous wood distillation plants in all countries, erected by the author and the firm of F. H. Meyer.

In his article Mr. Campbell speaks of the good prospects of the manufacture of acetic acid and methyl alcohol from the raw products of wood distillation. On the basis of my personal experience in wood distillation plants in all countries (United States, Canada, Germany, Hungaria, France, Belgium, Sweden and Russia), I confess that I cannot share this optimism. In all these countries experience has shown that refining plants for the production of methyl acetate, acetone, methyl alcohol, allyl alcohol from crude wood naphtha are profitable only on a very large scale.

There are very few wood distillation plants with an output large enough to operate a refining plant with profit.

In the United States, in Canada, in Hungary, briefly, in all countries with sufficiently large output, there are central refineries. These can be operated with much greater profit, as they are better able to utilize the intermediate products; further, the material losses are less; they can adapt the product better to the needs of the consumers with respect to quality; the expense of labor and fuel is less, and there are general commercial reasons with respect to freight expenses and market situation.

Moreover, at present, only continuous apparatus are used for the refining of crude wood naphtha. But these can give the best results only when operated on a large scale. Further, a crude wood naphtha refinery requires continuous control by a scientifically trained chemist, who can expect a salary which would be too much for a smaller refinery.

All these points seem to indicate that the refining of crude wood naphtha is not always certain to produce a reasonable profit. On the contrary, on the basis of practical experience, it may be said that only under specially favorable conditions

refineries can operate with a reasonable profit on a smaller scale.

Similar remarks may be made on acetic acid plants of smaller size. In this case, concentrated sulphuric acid is needed and its relatively low market value cannot stand considerable freight expenses.

Further, the present American producers of acetic acid, all of which use apparatus of the firm of F. H. Meyer, are also producers of sulphuric acid and this point should not be overlooked in considering the possibility of competition. Acetic acid manufacture also requires the employment of a scientifically trained chemist.

More favorable are the conditions for the manufacture of acetone. In the United States and Canada about 40,000 lb. of acetate of lime are treated per day by the process of the author for the manufacture of acetone and other raw materials are not required.

The above notes are intended to sketch briefly the features of wood distillation plants of modern type. In conclusion, it may be worth while to emphasize again that in case of a projected new plant or the rebuilding of an existing plant, it is necessary to first make sure that a reasonable profit can be obtained, and that the projected methods and apparatus really represent the latest developments of practice.

Convention of American Foundrymen's Association and Exhibition of Foundry Supplies.

The fifteenth annual convention of the American Foundrymen's Association and associated societies was held in Detroit in the week beginning June 6. What has been said of every convention of this association during the last years, was again true, namely, that the attendance was better than ever before and so was the exhibition of foundry supplies and machinery.

Mr. ARTHUR T. WATERFALL, in his presidential address, referred to the permanent mold and to the electric furnace as the most important problems of to-day before the foundrymen.

The secretary-treasurer, Dr. RICHARD MOLDENKE, emphasized that liberal mindedness and broad ideas of the individual foundrymen of this country had procured for the association its present high standing. He gave an account of two recent trips to Europe, and pointed out two noteworthy differences in foundry practice in this country and Germany. In Germany they make the castings entirely for the work intended, no matter whether the machine shop kicks about hard castings. If they want wear, they make the castings accordingly and do not allow any "excessive" machine shop costs to stand in the way. Dr. Moldenke thinks a little of this method would do good in this country, where we are often too much after soft castings.

The second point of difference is that the better brands of pig iron used in Germany are all of the low-silicon varieties, as they still believe in the necessity of importing metal that has the name for making strong castings. When the Germans will recognize that other pig irons will do the same, American and German practice will get closer together.

Two papers on oxy-acetylene welding were presented, one by Mr. STEELMAN STEPHENSON, of Detroit, on acetylene-oxygen repairs in the foundry; another, an illustrated lecture by Mr. HENRY CAVE on the methods of the Autogeneous Welding Equipment Company, of Springfield, Mass., who use the process of the Davis-Bournonville Company.

A shockless jarring machine was the subject of a paper by Mr. WILFRED LEWIS, of Philadelphia.

The premium system of compensation in the foundry was heartily recommended in a paper by Mr. W. J. POWER.

Rejected steel castings in steel foundries were the subject of a paper by Mr. SAMUEL D. I. EMERSON, who has reached the conclusion that in steel foundries rejections are principally due to accidents or to inefficient methods in core or molding

departments. Ninety per cent of rejections are mechanically avoidable, without reference to metallurgical difficulties. With adequate record, systematic and continuous attention rejections can be reduced to an average of 3 per cent. In order to secure the continuous attention required, a separate organization with pay largely dependent on results is most effective.

Mr. P. KREUZPOINTNER presented the report of the committee on industrial education. He emphasized the necessity of recognizing the social side of the problem of industrial education by the business men of this country. If our poorer European friends and competitors do not hesitate to expend enormous sums for the extension of their splendid system of industrial schools, then our industries may likewise consider such sacrifices as a profitable investment. Since we are not in a position for some time to come to increase the compulsory education age to 16 years, the employers ought to unite in establishing a standard of admission to work, which standard the school is to furnish and certify to, on dismissal of the pupils. This would compel the school people to get out of their isolation and to study the educational questions, not only from the academic standpoint, but also from the social and industrial standpoint.

Thus, on the pupils leaving school properly equipped we have to deal with two classes—the one class which cannot or will not go to school any longer, the other class going to school two years more. The former, going to work, ought to be taken care of by an arrangement between the school and the employer, the latter sending his employees under 16 years of age to school during a given number of hours during the week, or have them instructed in the shop, if desired. This is where the employer must make sacrifices. An outline of what should be required was given.

Dr. R. MOLDENKE presented a paper on specifications for foundry coke. This is printed in full elsewhere in this issue.

Mr. BENJ. D. FULLER discussed in a general way the methods of raising the efficiency of a foundry. Mr. THOS. D. WEST spoke on the personal equation in accidents. Foundry transportation methods were discussed in two papers by Mr. DAVID GAEHR and Mr. A. W. MOYER.

Dr. JOHN JERMAIN PORTER, for the committee on standards for iron castings, presented a paper on "chemical standards for iron castings." It is probably the largest collection of analyses of iron castings ever gathered into one table. Naturally, it cannot be abstracted.

Before the brass founders, Mr. E. S. SPERRY spoke on fluxes as applied to the brass foundry; Mr. C. R. STEVENSON on brass foundry costs and statistics; Mr. JESSE L. JONES on the selling of brass foundry refuse; Dr. JOS. W. RICHARDS on the electric power required to melt brass, bronze, etc.; Dr. J. J. PORTER on co-operative courses in metallurgy; Mr. CHARLES T. BRAGG on modern foundry progress; Mr. HUGH McPHEE on mounting patterns on molding machines; Mr. C. P. KARR on analysis of lead in brass alloys; Dr. W. F. HILLEBRAND on standard samples of brass, and Mr. H. M. LANE on magnesium for deoxidizing aluminium alloys.

Major JOSEPH T. SPEER, of the Pittsburgh Valve, Foundry & Construction Company, of Pittsburgh, is the new president of the American Foundrymen's Association. Mr. N. K. B. PATCH, of the Lumen Bearing Company, of Toronto, Canada, is the new president of the American Brass Founders' Association. Mr. GEORGE R. RAYNOR, of the Carborundum Company, of Niagara Falls, is the new president of the Foundry and Manufacturers' Supply Association. Mr. ROBERT B. THOMSON, of the Pitts Company, of Buffalo, is the new president of the Associated Foundry Foremen.

Exhibition.

As on former occasions, the chief interest centered in the exhibition. This was greater, more instructive, and more interesting than ever. We can give here only a few of the most notable exhibits.

The **Monarch Engineering & Manufacturing Company,** of Baltimore, exhibited a complete line of furnaces, operated by oil or gas, for the heating and melting of non-ferrous metals, ladle heaters, bolt heating furnaces, blowers, etc.

The **Rockwell Furnace Company,** of New York, showed their oil, coal or gas-fired melting furnaces, special burners, and complete furnace equipment.

The **Hawley Down Draft Furnace Company,** of Chicago, exhibited their line of melting furnaces, including the Schwartz furnace, the Reyellec furnace, etc.

The **J. D. Smith Foundry Supply Company,** of Cleveland, exhibited brass and aluminium melting furnaces, core ovens, sprue cutters, molding machines, and sand-blast equipment.

The **Anthony Company,** of New York City, showed a fuel-oil crucible tilting furnace for non-ferrous metal melting in operation; also high and low-pressure oil burners, etc.

The **Hauck Manufacturing Company,** of Brooklyn, N. Y., showed several types of oil-burning appliances, portable heaters, skin drying burners, cupola lighters, kerosene torches, forges of various types, etc.

Mr. **J. B. Wise,** of Watertown, N. Y., showed melting furnaces for non-ferrous metals.

The **Kroeschell Bros. Company,** of Chicago, Ill., exhibited their Kroeschell-Schwartz tilting and stationary crucible furnaces, ladle heaters, etc.

The **American Blower Company,** of Detroit, Mich., exhibited an electrically operated cupola blower in operation; further electric forge blowers, ventilating and heating apparatus, etc.

The **Detroit Foundry Supply Company,** of Detroit, Mich., had a general exhibit of foundry supplies and equipment, including brass furnaces, oil burners, core ovens, plating dynamos and platers' supplies of a very diversified character, molding machines, etc.

Another general exhibit of foundry supplies and equipment was made by the **Central Foundry Supply Company,** of Columbus, Ohio.

Resting rooms for foundry men with general foundry supply exhibits had been fitted up in the booth of **T. P. Kelly & Co.,** of New York City, and in the booth of the **S. Obermayer Company,** of Cincinnati, Ohio.

Walter MacLeod & Company, Cincinnati, Ohio, exhibited melting furnaces, oil burners, drying molds, sand-blast machines, etc.

The **J. W. Paxson Company,** of Philadelphia, Pa., exhibited a general line of foundry supplies, cupolas, sand blasts, magnetic separators, overhead tram rails, etc.

The **Whiting Foundry Equipment Company,** of Harvey, Ill., showed photographs of foundry equipment, cupolas, ladles, elevators, turntables, core ovens, cranes and various labor-saving devices for foundries.

The exhibit of the **Northern Engineering Works,** of Detroit, Mich., included the Newten cupola, a working model of a traveling crane, trolley cranes, hoists, foundry ladles, tramway systems, foundry elevators, etc.

The **Detroit Hoist and Machine Company,** Detroit, Mich., exhibited pneumatic hoisting machinery.

Victor R. Browning & Company, of Cleveland, Ohio, exhibited hoisting and conveying machinery for foundry use.

The **Ingersoll-Rand Company,** of New York City, exhibited compressed-air appliances for foundry use, air compressors, pneumatic sand hammers, chipping hammers, drills, motor hoists, etc.

The **Q. M. S. Company,** of Plainfield, N. J., exhibited air hoists, hand-power traveling cranes, trolleys, steel foundry saws, etc.

The **Calumet Engineering Works,** of Harvey, Ill., exhibited tumbling mills, ladles, trolleys, cupolas.

The **Hanna Engineering Works,** of Chicago, Ill., exhibited Rathbone multiple molding machines, screen shakers, mold dryers, etc.

The **Thomas W. Pangborn Company,** of New York, exhibited high-pressure sand-blast system, sand-separating systems, sand dryers and air separators.

The **Standard Sand and Machine Company,** of Cleveland, exhibited machinery for mixing and handling sand.

Frederick B. Stevens, of Detroit, Mich., exhibited foundry facings and general foundry and platers' supplies.

The **United States Graphite Company,** of Saginaw, Mich., had in their booth, which was fitted up as a resting room, an exhibit of plumbago foundry facings.

Stanley Doggett, of New York City, exhibited foundry facings and parting compounds, manganese dioxide, ferromanganese, ferrosilicon, etc.

The **Joseph Dixon Crucible Company,** of Jersey City, N. J., exhibited a complete line of graphite products, including crucibles, muffles, retorts, stirrers, foundry facings, etc.

The **Jonathan Bartley Crucible Company,** of Trenton, N. J., had a large exhibit of standard and special crucibles, and other foundry supplies made from graphite.

The **Carborundum Company,** of Niagara Falls, exhibited carborundum products, like grinding wheels, carborundum cloth and paper, also carborundum fire sand, which is used as a refractory.

The **Detroit Testing Laboratory,** of Detroit, Mich., showed electric furnaces for laboratory work, photomicrographs of metals, and charts giving the results of chemical and physical tests of metals.

Rogers, Brown & Company, of Cincinnati, Ohio, exhibited samples of pig iron, coke, fluorspar, and showed an interesting map exhibiting with electric flash lamps of different colors their sales offices, the blast furnaces, and the coke properties in which the company is interested.

S. Birkenstein & Sons, of Chicago, Ill., exhibited samples of ingot copper, pig lead, spelter, block tin and general brass founders' alloys.

The **Michigan Smelting and Refining Company,** of Detroit, Mich., exhibited yellow and red brass ingots and sample castings; also solders, babbitt metals, and various new nonferrous metals or alloys.

Hickman, Williams & Company, of Chicago, Ill., exhibited samples of pig irons, ferromanganese, ferrophosphorus, ferrosilicon, and silico spiegel.

The exhibit of the **Goldschmidt Thermit Company,** of New York City, included materials and appliances for the application of the thermit process in foundry practice; especially titanium-thermit cans for purifying molten iron and steel; nickel thermit for the treatment of cast iron; thermit for welding and repairing of flaws in castings. Demonstrations of the use of thermit in foundry practice were made. The exhibit of this company also included samples of the different metals and alloys made by the aluminothermic reaction, especially chromium, manganese, ferrotitanium, manganese-copper, manganese-zinc, and manganese-tin, all of these being free from carbon.

Analyzed Samples of Bureau of Standards.

The Bureau of Standards, Washington, D. C., announces that it is ready to distribute the following analyzed samples:

No. 6a. Iron D, replacing the original D of the American Foundrymen's Association.

No. 24. A vanadium steel with between 0.1 and 0.2 per cent vanadium. At present only a provisional certificate of analysis can be issued with this steel.

No. 25. A manganese ore, for total manganese and available oxygen.

No. 26. "Crescent" iron ore, for alumina, lime and magnesia.

No. 27. "Sibley" iron ore, for iron, silica and phosphorus.

No. 28. "Norrie" iron ore, for manganese (low).

A new schedule of fees goes into effect July 1, 1910, and thereafter the only discount allowed will be 10 per cent on four or more samples, alike or different.

Two new circulars relating to the bureau's standard samples have been prepared for free distribution. Of these, Circular 25 contains general information regarding all samples, and Circular 26 relates to the methods used by the bureau chemists and many others in analyzing the above manganese and iron ores.

American Peat Society.

The fourth annual meeting of the American Peat Society will be held in Ottawa, Canada, on July 25, 26 and 27.

Through the liberality of the Canadian Government and the great efforts of the president of the American Peat Society, Dr. Eugene Haanel, director of Mines of the Department of Mines, a very attractive program has been arranged.

The Canadian Government has a fuel testing station at Ottawa, Canada, which is equipped for the testing of peat only for power purposes. The plant consists of a double-zone peat-gas producer, wet coke scrubber, tar filter and dry scrubber, a four-cycle, single-acting Koerting gas engine of 60 hp, a 50-kw direct-connected Westinghouse direct-current generator, and a resistance stand for absorbing the generated energy. The plant will later be utilized for running the machinery of a metallurgical laboratory.

The peat for the operation of the producer is supplied from the peat bog owned by the Mines Branch, Department of Mines. This bog is situated at Alfred, on the Canadian Pacific Railroad, near Caledonia Springs, about 40 miles distant from Ottawa, and has an area of about 300 acres. The Anrep process for manufacturing air-dried peat is employed, the plant installed having a daily capacity of about 30 tons of air-dried peat.

Both the peat fuel testing plant at Ottawa and the peat plant at Alfred will be visited by the American Peat Society during the meeting. The professional program of papers is quite full, papers having been announced by Dr. E. Haanel, Mr. A. Dobson, Mr. A. Anrep, Jr., Dr. J. H. Pratt, Dr. T. A. Mighill, Mr. F. J. Bulask, Mr. E. V. Moore, Mr. F. H. Nagel, Prof. C. A. Davis, Dr. J. McWilliam, Mr. P. Heseltine, Mr. M. Joltz, Mr. J. C. Schumacker, Prof. R. H. Fernald, Mr. R. Schorr, Mr. O. E. Moulton, Mr. J. H. Van Glahn, Mr. William Eichoff, Mr. J. N. Hoff, Mr. R. Runson, Mr. J. Bordollo and Dr. O. Zwingenberger.

The hotel headquarters are at the Russell House. The chairman of the local reception committee is Mr. Arthur J. Forward, Ottawa, Canada.

Electrolytic Cyanidation and Amalgamation.

By Elmer Ellsworth Carey.

Inventions and processes are frequently announced many years before the world is ready for them. The electrolytic cyanide process was described by Rae in 1867, but to-day only one part of that process—electrodeposition—is in use. The electrolytic system of separating and refining metals is now rapidly superseding all other metallurgical systems, and the electrochemical system will before long be used extensively in the metallurgy of gold. By the use of electricity the percentage of gold won will be increased, the cost of extraction diminished, and the area of mining operations greatly enlarged.

As soon as the principles of electrochemical metallurgy taught by the various colleges, institutes, technical schools, etc., become better recognized in practice, important modifications and changes will be made in milling operations. As cyanidation made slow progress till a new generation of engineers had been trained in the use of the chemical system, so the electrochemical metallurgical methods will languish till the schools have supplied the necessary electrochemical engineers.

It has been known theoretically for 40 years that the amalgamating action of mercury and the energy of the cyanide

solution could be increased by the introduction of certain electrochemical conditions, but many difficulties had to be overcome before this knowledge could be used in a practical manner. Within the last 10 years the science of electrochemistry has been placed on a firm foundation, and the troubles of the early inventors have been surmounted. Low-voltage generators are now manufactured that produce the required current at a low cost; by the use of Acheson graphite, all anode trouble disappears; and the time of treatment has been reduced to a few hours, while in some systems the values pass into solution and are deposited simultaneously.

Metallurgical science took a great step forward when the chemical (cyanide) system was introduced, and the next important step will be the adoption of the electrochemical system. Many troubles, obstacles and difficulties of cyanidation disappear when electrochemical conditions are introduced; values which ordinarily are refractory readily pass into the solution; a weaker solution than usual may be used; the solution is constantly regenerated; the loss of cyanide is reduced; the values are automatically recovered in the form of amalgam; the refining processes, with their attendant losses, are unnecessary, and the cost of installation greatly reduced.

In the *Mining World*, Jan. 23, 1909, may be found a description of the Parks electrocyanide process. The process here described is extremely simple, effective and inexpensive, and has been found successful with ores which are not amenable to the usual cyanide treatment. In the works on cyanide practice by Bosqui, Wilson, Smart and others will be found descriptions of other systems of electrocyanidation, all of which have similar basic features, differing only in detail, modes of construction, etc. Some years ago electrocyanidation was successfully used for several months at the Delamar Mine in Idaho, and a description of the process there used will be found in Bosqui's work.

The recovery of values from a cyanide solution (after the usual leaching process) by electrodeposition has proved successful in many mills in South Africa; the Siemens & Halske process of electroprecipitation was introduced several years ago on the Rand, and despite the primitive apparatus used, the high cost of installation, and the 5 per cent royalty, many companies have been using the process for a number of years.

To-day the special process in question is obsolete, as far better methods are known; Acheson graphite is much better for anodes than the clumsy anodes used in the German system, and amalgam plates can now be used instead of the lead cathodes. With the introduction of discoveries, methods, machinery, and accessories which have been placed at our disposal by American inventors and investigators in the electrochemical field, the time has now arrived when the art of cyanidation can be completely revolutionized. Some of the advantages that may be expected from the improved system are mentioned in an article of electrochemical amalgamation and cyanidation in the *Engineering Magazine* for December, 1909, as follows:

"Turning to electrochemical lixiviation, the claim is made that under electrochemical conditions the cyanide solution is much more active than usual; values pass into the solution which are lost in the ordinary systems; the time of treatment is reduced and the extraction increased, while there is a decrease in the operating cost; precipitation by electrodeposition goes on simultaneously with the leaching process and the usual troubles are eliminated; a molecule of cyanogen after taking up a particle of gold, is released as nascent cyanogen, having greatly increased affinity for gold and silver. The solution is constantly regenerated, the loss of cyanogen being negligible. The larger particles of gold are amalgamated directly without passing into the solution. In electrocyanidation the pulp and solution are agitated in a shallow tank having the bottom covered with quicksilver (cathode); revolving stirrers cause the pulp to travel with a circular motion over the quicksilver for an hour or two; this is equivalent to passing the pulp over several miles of mercury surface. During this period the values are amalgamated directly, or pass into the solution and are precipitated by electrolysis, the extraction averaging 97 per cent. All ores, so far as known, are capable of successful treatment by means of electrocyanidation."

Regarding electroamalgamation, it may be said that experimental work relating to electrochemical amalgamation began in 1859, the first patent being issued to Henry Brevoost, of San Francisco; some 75 patents for various electro-amalgamating devices have been issued by the United States Patent Office, but only within the last three years have machines been constructed which meet the requirements of mining men. The various engineering difficulties have been overcome and an electro-amalgamation device is now constructed which is very simple, inexpensive, has large capacity, is easy to operate, skilled attendance not being necessary, and which extracts all values not encased, regardless of refractory elements which may be present in the gangue.

Such a device works equally well with mill pulp or screened placer material; in placer operations all rusty, coated and black sand values, as well as platinum, are amalgamated; fine, float and beach gold is readily amalgamated in the electrolytic device. Those who wish an authoritative statement as to the value and scope of electrochemical amalgamation are referred to a paper by Warnford-Lock mentioned below, and also to the patent issued to Sir William Crookes.

Those who wish to further investigate electrocyanidation are referred to an article by John R. Parks in the *Mining World*, Jan. 23, 1909; an article on "Electrochemical Amalgamation and Cyanidation," in the *Eng. Mag.*, December, 1909, and the following books: "Cyanide Process," by E. B. Wilson; "A Handbook of Practical Cyanide Operations," by Wm. H. Gage; "Practical Notes on the Cyanide Process," by F. L. Bosqui; "Electric Smelting," by Dr. W. Borchers, and "Cyaniding Gold and Silver Ores," by H. Forbes Julian and Edgar Smart.

Additional references for data regarding electroamalgamation are as follows:

"Electrochemical Amalgamation," C. G. Warnford-Lock, E. M., M. L. I. M., Vol. 1, page 205, *Proceedings* of the (London) Institution of Min. and Met.

United States patent No. 462,535, issued to Sir William Crookes for an electrolytic amalgamating process; the following numbers refer to other United States patents covering devices for saving gold by electrolytic methods: 328,532, 307,081, 342,421, 492,711, 590,524, 947,957, 947,958, 757,557.

Other articles on electro-amalgamation have appeared in the *Mining Journal* (London), May 15 and June 12, 1909; *Mining World*, March 27, April 17 and June 5, 1909; *Pacific Miner*, April and August, 1909, also February, 1910; *Electrochemical and Metallurgical Industry*, May, 1909; METALLURGICAL AND CHEMICAL ENGINEERING, March, 1910, page 116 and 157; *Scientific American*, Aug. 7, 1909; *Scientific American Supplement*, Nov. 6, 1909; *Mexican Mining Journal*, October, 1909, and January, 1910.

SAN JOSÉ, CAL.

Commission of Conservation of Canada.—The Commission of Conservation of Canada (Hon. Clifford Sifton, chairman, James White, secretary) has just issued the report of the first annual meeting held at Ottawa from January 18 to 21, 1910. It is a very interesting volume of 216 pages. Besides the inaugural addresses by Hon. Clifford Sifton and by the Governor-General of Canada, Earl Grey, it contains papers by Dr. B. E. Fernow on scientific forestry in Europe, by Dr. J. W. Robertson on the conservation of agricultural resources, by Dr. Eugene Haanel on possible economies in the production of minerals of Canada, by Hon. Frank Cochrane on the conservation of the natural resources of Ontario, by Hon. Adam Beck on the conservation of the waterpowers of Ontario, by Mr. C. E. Contlee on the water wealth of Canada, etc.
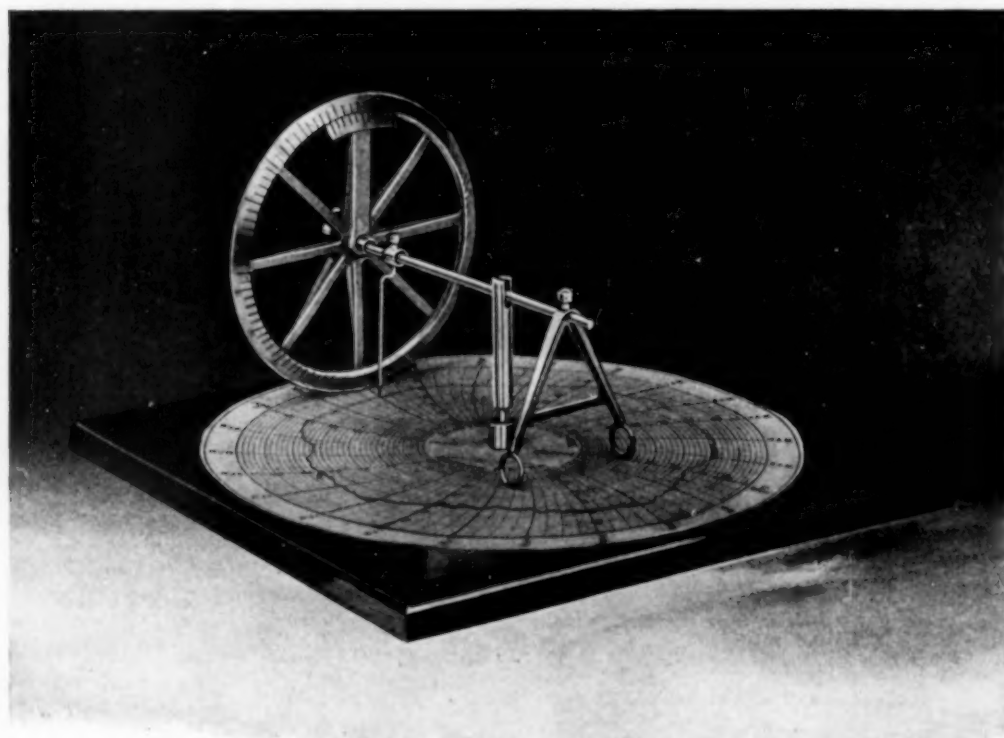
Averaging Instrument for Circular Chart Records

The Bristol Company, of Waterbury, Conn., have for the last 20 years made a specialty of manufacturing automatic recording instruments for pressure, temperature and electricity. As this class of recording instruments using circular charts have come into general use there has developed a demand for a simple device to quickly determine the average of the record made on such charts, and the integral value for the whole 24 hours, or for the time covered by the record.

This long-felt demand has been filled by the instrument illustrated herewith, which is based upon a fundamental plan as worked out and patented by Prof. W. F. Durand, of Stanford University, and is constructed in accordance with a novel design recently patented by Wm. H. Bristol, president of the Bristol Company.

The instrument can be applied for averaging records of any kind on circular charts having uniform graduations, as, for instance, records of watts, amperes, temperature, pressure, etc.

AVERAGING INSTRUMENT FOR CIRCULAR CHART RECORDS.

Recording instruments equipped with circular charts are therefore made available for a number of applications for which it was previously thought necessary to use the instruments recording on straight lines or strip record charts. Recording differential pressure gages are coming into use for measuring velocities and volumes of liquids, air or gas flowing in mains, and this integrating device will prove of value for quickly obtaining total volumes of flow for any given period of time.

The simple construction of the instrument is shown in the accompanying illustration. A wooden base with a metal socket is provided for supporting and centering the chart. The socket holds a rotatable pin with a vertical slot at the top to receive the bar which carries the integrating tracer point and triangular support. The vertical groove in the rotatable pin allows the integrating wheel to roll on the chart with uniform pressure due to its own weight.

The integrating wheel is 6 in. in diameter, the rim being graduated into 100 numbered equal spaces, and is fitted with a vernier, which makes it possible to easily read with the naked eye to one-tenth of one division on the integrating wheel. The

wheel is of such large size that it is not necessary to supply any counting device for the number of revolutions. The number of complete revolutions cannot be more than two, even for a record of maximum size on the large 12-in. charts.

To operate the instrument the thumb and forefinger of one hand are applied to the base of the triangular support, which is moved radially, so as to cause the tracer point to continually follow the record curve, while the chart is turned with the other hand.

By referring to a line plotted on a sheet of cross-section paper furnished with the instrument, for the particular record curve that is to be measured, the total reading for the entire 24 hours may be taken off directly.

A full explanation of the theory upon which the operation of the instrument depends has been given by Professor Durand in a paper presented at the New York meeting of the American Society of Mechanical Engineers in 1908. This may be briefly summarized as follows:

In applying the instrument it is necessary to have a uniform radial scale, from which it follows that equal increments in the length of the radius correspond to equal increments in the watts, amperes, temperature or whatever quantity is measured.

The integrating wheel being carried at right angles to shaft passing through the center does not turn and give a reading when the tracing point is moved on a straight radial line, but if the tracing point is made to follow a record, the integrating wheel will revolve and the amount of the revolution will correspond to the total of the circumferential elemental components of the record curve, the radial elemental components of the record having no turning effect on the integrating wheel.

As the lengths of the arcs of concentric circles for given angles or for the entire circumference are proportional to their radii, it is evident that the amount of turning of the integrating wheel, and the reading obtained thereon, will be proportional to the average radius of the record traced.

The instrument is adapted for integrating charts with either straight or curved radial time arcs. The correction necessary for radial time arcs which are curved may be made, after completely tracing the record, by returning the tracing point to a point on the chart having the same radius as the starting point, the movement of the tracing point being along an arc corresponding to the curved radial time arcs.

American Society of Civil Engineers.—The forty-second annual convention of the American Society of Civil Engineers was held in Chicago from June 21 to 24. A very enjoyable series of entertainments and excursions had been arranged, to Gary, Ind., to the drainage canal and its power plant and various others. Illustrated lectures were given by Mr. A. Noble on the New York tunnel of the Pennsylvania R. R., and by Mr. J. W. Smith on the New York water supply system.

NOTES.

American Iron and Steel Institute.—The first regular meeting of the American Iron and Steel Institute, organized in 1908, was held on May 27, 1910, at the Waldorf-Astoria in New York. The first annual meeting will be held in October and will last two days. The officers elected are: E. H. Gary, president; P. Stackhouse, W. L. King, Charles M. Schwab, vice-presidents, Edward Bailey, treasurer; W. J. Filbert, secretary.

Mica Production in the Custer district of the Black Hills, South Dakota, is being carried on by the Westinghouse Company and the Chicago Mica Company. The former ships its products to Denver mills for dressing.

The Steptoe Smelting Company, operating one of the large copper concentrators and smelters in the Ely district, Nevada, is reported to be running all (eight) sections of the concentrators, the full battery of the McDougal roasters (16) and three out of its four reverberatory furnaces.

The Cananea Consolidated Copper Company, in Cananea, Sonora, Mexico, is increasing its roasting plant and has placed an order with Allis-Chalmers Company for six McDougall furnaces. These are exactly like the four furnaces which they now have in service. The furnaces have an outside diameter of 18 ft. 6 in. and are 32 ft. high. They are the six-hearth type.

Census of Electric Central Stations.—According to figures just issued by the Census Bureau of the Department of Commerce and Labor, the number of commercial and municipal plants increased from 3620 in 1902 to 4714 in 1907, which is an increase of 30.2 per cent. The application of the same rate of increase to the estimated number (50,000) of isolated plants in 1902 gives an estimate of 65,000 for 1907. Of the 4714 central stations reported in 1907, 227 had no generating equipment, while 113 had more than one power plant, 337 generating plants being reported from these 113 stations. There were in 1907 upward of 30,000 individuals, companies, corporations and municipalities, exclusive of isolated electric plants, which reported the generation or utilization of electric current in commercial enterprises. These industries represent an outstanding capitalization of $6,209,746,753, of which $1,367,338,836 is credited to electric central stations, $3,774,722,096 to electric railways, $814,616,004 to commercial or mutual telephone companies, and $253,019,817 to telegraph companies.

Electric Heating for Compressed Air Pumps.—The possibilities of electric devices for the most varying purposes in connection with metallurgy seem to find attention in Shasta County, California, the classical district in which the electric furnace has first been tried in this country for iron-ore reduction. The latest application of electric heating is due to Mr. H. A. Sutliffe, of the Bully Hill copper mines, who uses a simple electric heating device in connection with pumps operated by compressed air. The object of heating is simply to raise the air pressure just before the compressed air is utilized. It is claimed that this is so successfully done that each electric heater which is operated with a current of 40 amp from the 110-volt supply circuit and therefore consumes 4.4 kw, saves something like $6 a month. The device consists of a resistance-wire coil around the air pipe, provision being made, of course, for proper insulation and for disconnecting the resistance coil when the pump in question is not working.

Gary, Ind.—The first plant of the American Sheet & Tin-plate Company, at Gary, Ind., on the lake shore just west of the Indiana Steel Company property, is expected to be finished by March, 1911, with 2000 employees. The American Bridge Company is also well along with its new Gary shops. Both plants will use about 30,000 kw. This power will be obtained from the great gas-engine driven generating stations at Gary and South Chicago (working in parallel) of the U. S. Steel Corporation.

PERSONAL.

Mr. **Geo. F. Duck,** formerly consulting engineer of Pittsburgh, has succeeded Mr. R. L. Herrick as Western editorial representative of *Mines and Minerals,* with headquarters in Denver, Col.

Dr. **W. A. Hendry,** manager of the Hendryx Cyanide Machinery Company, of Denver, has removed to New York City, where the business of his company will in the future be conducted. The new offices of the company are at 107-109 William Street.

Mr. **R. L Herrick,** who has been the Western representative of *Mines and Minerals* for several years, has resigned his position and will engage in business near Cincinnati, Ohio.

Mr. **E. M. Johnson** recently accepted a position as assistant superintendent with the Mineral Point Zinc Company, Depue, Ill.

Mr. **J. M. McClave,** metallurgical engineer of Denver, Col., is engaged in experimental work near Fairview, Col.

Mr. **Charles M. Palmer,** formerly in charge of the B. & M. laboratory in Butte, Mont., has been transferred to the smelter at Great Falls, Mont., where he will be engaged in work on the smoke and fume problem.

Mr. **H. F. Parshall,** the distinguished consulting engineer of London, has received the degree of Doctor of Science from Tufts College.

Dr. **Jos. W. Richards,** of Lehigh University, has sailed for Europe to attend the International Mining and Metallurgical Congress in Düsseldorf, but will be back early in July.

Mr. **J. B. Risque,** of Salt Lake City, who has been general manager of Utah Consolidated Company for the past three years, has resigned his position, to take effect Sept. 1. Mr. Risque will continue to engage in mining and metallurgy after his resignation, with headquarters in Salt Lake City.

Mr. **W. J. Sherwood,** manager of the Rambler Copper & Platinum Company, Holmes, Wyo., recently spent a few days in Denver, en route to the company's head offices in Chicago.

Mr. **W. G. Swart,** representative of the American Zinc Ore Separating Company, with headquarters in Denver, recently spent several weeks in Montana on professional business.

Mr. **Theo. F. Van Wagenen,** who has been spending some months at Zacatecas, Mexico, on professional business, has returned to Denver.

Dr. **G. B. Waterhouse** has been appointed metallurgist of the Lackawanna Steel Company, to succeed Mr. H. Cook.

Mr. **R. H. Wolff,** of New York, representative of the Héroult electric steel process in this country, sailed for Europe on the *Mauretania* on June 22, to be abroad for about two months.

NEW BOOKS.

Albe, E. E. Fournier d'. The electron theory. A popular introduction to the new theory of electricity and magnetism; with a preface by G. Johnstone Stoney; with front and diagrams in text. Third edition. 354 pages. Price $1.25 net. New York: Longmans, Green & Co.

Moore, F. J. Outlines of organic chemistry. A book designed especially for the general student. 324 pages. Price $1.50, net. New York: John Wiley & Sons.

Sexton, A. H. Fuel and Refractory Materials. 364 pages. Price $2.50 net. New York: D. Van Nostrand Co.

Sharp, J. Modern Foundry Practice. Second edition revised. 769 pages. Price $8.00. New York: Spon & Chamberlain.

BOOK REVIEWS.

Modern Assaying. By J. Reginald Brown. Edited by **F. W. Braun.** 145 pages. Cloth. Price, $1.50. Philadelphia: J. B. Lippincott Company.

The sub-title of this book describes it as "a concise treatise describing latest methods and appliances," but it may be said to be more complete in the latter respect than in the former. It illustrates the great difficulty of presenting a technical subject in a "popular," or untechnical manner, at the same time confining its descriptions of apparatus to that of one manufacturer. Technically, the volume cannot have as great a value as some other treatises on the same subject, although it may find acceptance with a class of operators who are not deeply versed in the technical refinements of their work, and who are content to follow directions without a knowledge of the underlying reasons, causes and effects. Methods are given for the fire assay of gold, silver and lead; electrolytic assay for lead and copper, and wet methods for lead and copper. The retorting of amalgam is also described.

*   *   *

Tables for the Determination of Minerals. By **Persifor Frazer**, D.Sc., and **Amos P. Brown**, Ph.D. 125 pages. Flexible leather. Philadelphia: J. B. Lippincott Company, Price, $2.50 net.

This is the sixth edition of the authors' translation of Prof. Albin Weisbach's *Tabellen*. The original tables were compiled by Dr. Weisbach for the use of his students at Freiberg, and were based on determination from outward appearance and physical characteristics. While a complete classification of minerals cannot be made on this basis, nevertheless it has been found more convenient in general use than a more elaborate system based on chemical composition.

Weisbach divided minerals into three systems, viz.: those of metallic luster, those of non-metallic luster which give a colored powder, and those of non-metallic luster and colorless streak. These three systems have subdivisions so that it is comparatively easy to classify a mineral in a short time by observing closely its physical characteristics.

The translators have retained Weisbach's system and have added to it accurate chemical formulae of the minerals. Dana's system of nomenclature, Mohs' scale of hardness and von Kobell's scale of fusibility are used throughout the work.

The volume should be of great value to students and to those who find it necessary to identify minerals in field work. An excellent index adds to the value of the work.

*   *   *

Metallography. By **Cecil H. Desch**, D.Sc. (Lond.), Ph.D. (Wurzb.), Graham Young, Lecturer in Metallurgical Chemistry in the University of Glasgow. London and New York: Longmans, Green & Company. Price, $3.

This book is divided into 18 chapters which deal with the following headings: Introduction; The Diagram of Thermal Equilibrium; Solid Solutions or Mixed Crystals; Ternary and More Complex Systems; Metals which are Only Partially Miscible in the Liquid State; Practical Pyrometry and Thermal Analysis; The Preparation of Micro-Sections; The Microscopical Examination of Prepared Sections; The Crystallization of Metals and Alloys; Undercooling and the Metastable State; Diffusion in the Solid State; The Physical Properties of Alloys; Density—Thermal Expansibility—Hardness—Electrical Conductivity—Thermo-Electric Power—Magnetic Properties; Electromotive Force and Corrosion; The Construction of the Equilibrium Diagram; The Molecular Condition of Metals in Alloys and the Nature of Inter-Metallic Compounds; The Plastic Deformation of Metals and Alloys; The Metallography of Iron and Steel; The Metallography of Industrial Alloys.

This book sets forth very clearly our present knowledge of metallography, which is defined as a study of the internal structure of metals and alloys and of its relation to their composition and to their physical and mechanical properties. As it is one of

the text-books of physical chemistry edited by Sir William Ramsay, it naturally takes up the study from that standpoint.

Starting with the diagram of thermal equilibrium, the basis of which is the freezing point curve, the author, by clear steps, works out the different cases and merely touches upon the phase rule and shows its application to alloys. The discussion of the ternary systems is exceedingly well written. Under the heading of practical pyrometry and thermal analysis, we have given us in 40 pages a condensation of the methods in use and the instruments for the autographic registration of curves. It would seem to us that this chapter was rather too condensed for the ordinary student; however, the references given cover the subject very thoroughly.

The chapter on the Crystallization of Metals and Alloys is exceedingly well written and the microphotographs illustrating it are very happily chosen. The chapter on the physical properties of alloys, covering density, thermal expansion, hardness, conductivity, etc., covers the ground very thoroughly and we can gather a very clear idea of what has been done and what remains to be done on this side of the subject.

The latter part of the book, dealing with the metallography of iron and steel, summarizes, in a very few words, our present knowledge of the subject and the book ends with an appendix consisting of tables of systems of which equilibrium diagrams have been published, the probable formulae of inter-metallic compounds present have been added. The table is divided into: eutectiferous series, isomorphous series, the metals only partially soluble in liquid state; series forming a single compound, complex series in which two or more compounds are formed, and systems which have not been completely investigated, and, lastly, the ternary systems.

On reading through this book, one is immediately struck by the feeling that Mr. Desch is familiar with his subject from the practical standpoint and not merely from reading the researches of others. He speaks with authority and, therefore, what he says is convincing. In addition, the style is so straightforward and simple that the book will be found both instructive and fascinating, not only to students, but also to practitioners who are interested in metallurgy in general, and alloys in particular. The book covers a very wide field, but covers it well, and Mr. Desch is to be congratulated in supplying a long-felt need, because there is no doubt that to-day metallography has become quite an important branch of study.

*   *   *

Fuel and Refractory Materials. By **A. Humboldt Sexton**, F.I.C., F.C.S., emeritus professor of metallurgy in the Glasgow and West of Scotland Technical College. New and revised edition. 364 pages, illus. Price, $2.50 net. New York: D. Von Nostrand Company.

There are 15 chapters, dealing with combustion; heating power of fuels; wood, peat, coal; charcoal, peat-charcoal, coke; coal washing; liquid fuels; gaseous fuel; recovery of by-products; furnaces for metallurgical purposes; supply of air to the furnace, removal of waste products, smoke, prevention of smoke; pyrometry; calorimetry; utilization of fuel; testing fuels; refractory materials, bricks, crucibles.

This is the second edition of Professor Sexton's book and while no essential change has been made in the scope and general arrangement of the book, yet many additions have been made to the chapters on coke ovens, gas producers, and pyrometry. Descriptions of modern forms of plant and apparatus have been added and some descriptions of older types of plant have been deleted.

The author says: "In spite of the progress which is being made in the adoption of the metric system in this country, it is unfortunately still necessary to use the British system of weights and measures. These have, therefore, been retained." But why not adopt a compromise and give both the metric units and the British units, perhaps the latter in parenthesis? Temperatures are given by the author both in degrees Fahrenheit and Centigrade. The book is well and clearly written.